THE MOVEMENT OF GASES IN LONGWALL COALFACE WASTES LIABLE TO SPONTANEOUS COMBUSTION

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Abstract

The increasing depth, distance from surface connections and falling quality of coal mined are factors that increase the risk of spontaneous combustion in working areas. The trend towards high capital investment, high output faces significantly raises the economic consequences of spontaneous combustion. Much work has been directed towards methods of identifying the liability of a coal to spontaneously combust under given conditions. The cost of prevention and combat of spontaneous combustion underground is high, however this work is carried out with little knowledge of the likely location of a heating in a coalface waste. This thesis investigates the airflow patterns in a coalface waste, with a view towards improving the use of prevention and combat methods.

The factors that affect the liability of a coal to spontaneously combust, and the methods of prevention and combat are discussed to provide a background to the subject area.

An investigation into the flow of nitrogen that was injected from one hole into a coalface waste as a combat measure was conducted. The results of this showed how the amount of nitrogen entering the waste depended on the rise and fall of the atmospheric pressure.

An attempt was made to develop a method of sampling gas from deep within the coalface waste. It proved impossible to sample further than 15m behind the face line. The results from this exercise are presented.

The finite element method was used to model the pressure distribution in the waste under differing boundary pressure and waste permeability conditions. A suggested area at risk from spontaneous combustion is presented.

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CHAPTER 1

INTRODUCTION AND RESEARCH OBJECTIVES

1.1 INTRODUCTION

Spontaneous combustion has long been recognised as a problem in underground coal mines (1). In recent years the exhaustion of the United Kingdom's reserves of high quality coals in shallow seams has led to the exploitation of lower quality reserves at greater depths, and locations underground remote from existing mine shafts.

The introduction of fully caved wastes, as opposed to the previous methods of stowed wastes or bord and pillar mining techniques, has resulted in the creation of a large void, filled with broken carbonaceous material, through which air flows, behind every face line. Coupled with this exploitation of deeper reserves there has been an increasing trend towards highly mechanised, high output, working faces. Heavy duty support and production machinery has been installed as part of a policy of concentrating output from

fewer faces.

These factors have resulted in raised air temperatures and humidities underground, which have been combatted by an increase in ventilating pressures and quantities (2).

These elements lead to a rise in the risk of spontaneous combustion, and greater severity of the economic consequences of any outbreak.

1.1.1 MECHANISM OF SPONTANEOUS COMBUSTION.

Spontaneous combustion can be defined as the self heating and slow combustion of coal and carbonaceous material initiated by the adsorbtion of oxygen. Coal will exothermically oxidize whenever it is exposed to the atmosphere. The rate of adsorbtion increases with temperature, and at temperatures above 40^{o}C adsorbtion of oxygen becomes a chemical reaction.

The chemical reaction above 40^{o}C is low temperature oxidation, between the coal and molecular oxygen, which proceeds over the following steps with increasing temperature.

i. The formation of solid coal/oxygen compounds.

ii. These may decay to form carbon dioxide, water and other stable carbon compounds.

iii. At temperatures above 100°C these break down into gaseous products.

iv. The result of this sequence is the breakdown of the aliphatic structures of coal.

The oxidation of coal is an exothermic reaction. If the rate of heat loss is equal to the rate of production, the reaction will not continue beyond the first stage. If, however, the rate of heat production exceeds the rate of loss, the temperature rises, and the rate of oxidation increases, leading to an increase in the rate of heat production. An accelerating, self sustaining process is started. This can only be stopped by the exhaustion of the combustible material, removal of oxygen from the reaction, or sufficient heat loss to prevent further oxidation.

The amount of heat generated is dependent on the rank of the coal, the lower the rank the greater the rate of heat generation.

Spontaneous combustion is governed by two groups of factors. One group, called the intrinsic factors, are the physical and chemical properties of the coal which define the oxygen adsorbtion and oxidation characteristics of a coal. The second group, the extrinsic factors, are the external factors brought about by mining the coal. These determine the supply of air to the coal, and the removal of heat from it. The ways that these factors affect the liability of a coal to spontaneous combustion are discussed in the following chapters.

1.1.2 PREVENTION AND COMBAT OF SPONTANEOUS COMBUSTION

The prevention and combat of spontaneous combustion underground is achieved by excluding oxygen rich air from the suspected site of a heating. The actual methods used are reviewed in Chapter 4. In some areas, such as the face start line, it is relatively easy to broadly identify the site of a heating, and take appropriate measures. However, when a heating is detected deep in the waste behind a face there is often little guidance available as to the location. This has led to the introduction in recent years of nitrogen injection into the waste. The theory behind this is that the oxygen content in the waste is reduced over a large area, sufficiently to inhibit combustion.

1.2 RESEARCH OBJECTIVES

Much research has been done into the factors that govern a coal's liability to spontaneous combustion. Work is continuing in this field towards defining quantitatively a risk index of a coal to spontaneous combustion. However the complex, and varying, physical and chemical constituents of coal, along with differences in the geological and mining induced factors, has made a precise quantification of the risk impossible. In the absence of a universal risk index the accent falls onto the prevention, detection and combat of heatings in areas known from previous mining experience to be of high risk.

There has been little work reported on the airflow conditions likely to initiate spontaneous combustion, or of the air movement in the waste that leads to spontaneous combustion. Nitrogen injection is being used in prevention and combat with little idea of where a heating might occur, or where the flow of nitrogen is going in the waste.

The objectives of the research were to more closely identify areas in the waste at risk to spontaneous combustion, and to improve the utilisation of nitrogen in prevention and combat.

The research was carried out in three stages. A face which was being salvaged and was having nitrogen injected as a precautionary measure, was chosen to investigate the behaviour of nitrogen once injected into the waste. The composition of the waste gas at increasing depth in the waste behind the face was examined during an attempt to develop a reliable method of monitoring the gas composition deep in the waste. Finally, the airflow in the waste for a number of different conditions was modelled using the finite element technique.

CHAPTER 2

PROPERTIES OF A COAL THAT AFFECT LIABILITY TO SPONTANEOUSLY COMBUST

2.1 INTRODUCTION

Coal is a complex substance, composed of varying proportions of organic and inorganic constituents.

For many years it was thought that the presence of iron pyrites in coal was the cause of spontaneous combustion. Experiments by Richters(3) showed that pyrites played little part, and the cause of heating is the oxidation of coal.

Subsequent work has shown that the oxidation potential of a coal bears a relation to the liability of it heating. Winmill(4), and other workers, showed that the capacity of a coal to absorb oxygen could be used as a guide to the self heating liability. Since low rank coals generally have a higher oxidation potential than higher rank, the rank of a coal is recognised as a guide to its liability to spontaneous combustion.

Efforts are continuing to analyse the effects that the differing constituents and properties of coals have on the spontaneous combustion liability. This will provide a method of classifying coals according to their risk factor.

Other factors apart from the coal itself have to be considered. The geological situation of a coal seam has an apparent effect on the liability of the coal to spontaneously combust, as does the physical structure of the coal. This chapter outlines the intrinsic factors that influence a coal's liability to spontaneous combustion.

2.2 THE COMPOSITION OF COAL

The constituents of coal have an obvious effect on its oxidation potential. Coal has a complex structure of inorganic and organic compounds. Because of this it has not been possible to single out constituents in the oxidation reactions. The influence of rank on the rate of oxidation is obvious, but the influence of the rank controlling constituents is not clear.

2.2.1 VOLATILES CONTENT

Low rank coals have a high volatiles content. It has been shown that coals with a high volatiles content are more reactive than others(5). Upon heating these coals evolve volatiles at a greater rate, and quantity, than coals of higher ranks. Light volatiles are evolved at lower temperatures. Theoxidise easily and

ignite at a low temperature. Even before combustion, the lighter volatiles undergo a highly exothermic reaction with oxygen. The evolution of volatiles may provide a method of detecting heatings at an early stage.

The rate of heat production is proportional to the rate of devolatisation, and in turn the build up of temperature.

2.2.2 MOISTURE CONTENT OF COAL

The moisture content of coal is a rank controlling factor, low rank coals have a high moisture content. There are differing views on the effect of inherent moisture on spontaneous combustion liability. In order for a coal to fire, all the water would have to be driven off. The evaporation of water would remove heat from the coal, thereby reducing the temperature and slowing the heating down.

The converse view is that a high moisture content will increase liability. The evaporation of water will wash off the inactive surface of the coal, while the drying out causes decrepitation, leading to the production of fresh, active surfaces.

The important parameter appears to be the difference in the moisture content between air and coal. When water evaporates off the coal there is a heat loss. When water condenses on the coal there is a heat gain. This is significant at low temperatures where the heat input from condensing water (2440KJ/KG) is greater than the heat produced by oxidation. A situation may be caused where there is a dry spot of coal, with fresh surfaces, surrounded by warm, wet

coal acting as insulation.

Hodges + Acherjee (6) showed that at 30 C, for a small coal sample, the heat generated by oxidation is far less than that removed by moisture loss.

2.2.3 ASH CONTENT

Ash comprises of all the inert matter in a coal. The presence of ash in a coal dilutes the combustible material, and so would reduce the liability. However it should be noted that waste dumps, composed largely of shale, burn. Some opencast mines have problems with the high walls, which are shale, burning (7).

2.2.4 PETROGRAPHIC CONSTITUENTS

Coals of the same rank do not always have the same liability to spontaneous combustion. There are two areas to be considered, lithotypes and macerals.

Of the lithotypes, vitrain is the most reactive. Both vitrain and fusain are soft and easily broken. This will produce fines. Clairain and durain are harder, and less reactive, having lower volatiles and a higher carbon contents.

Chamberlain(8) stated that the order of reactivity of the macerals is exinite, vitrinite,inertite.

Study of the petrographic constituents may aid in assesment of liability to spontaneous combustion.

2.2.5 OXYGEN CONTENT

Inherent oxygen content of the coal is a rank controlling factor. Oxygen is held in the coal as chemical compounds, mainly oxides, and as a small amount of gaseous oxygen adsorbed onto the coal. Experimental results have been inconclusive. No real correlation, or otherwise, has been shown between oxygen content and the rate of oxidation.

2.2.6 SULPHUR CONTENT

There are three forms of sulphur that occur in coal; pyrites, organic and sulphates. Of these, only the presence of pyrites is considered to be of any significance, but differing views are held.

The findings of Winmill (4) were that the liability of heating was raised if a high proportion of finely divided pyrites was present in the coal.

Pyrites are more reactive with air than coal, producing a much higher specific heat output. The different rates of thermal expansion, with pyrites increasing in size more than coal, promotes cracking of the coal, exposing fresh surfaces. When combined with water in air, iron pyrites undergo the exothermic chemical reaction:

$$2FeS_2 + 7O_2 + 2H_2O = 2FeSO_4 + 2H_2SO_4$$

Oxidation is a surface process for pyrites, which slows as ferrous sulphate is formed. Small, finely disseminated crystals will continue to oxidize at a higher rate for a longer period of time.

More recent findings by Schmit + Elder(9) show little correlation between the pyrites content of a coal and the spontaneous combustion liability.

2.3 STRUCTURE OF COAL

Accumulations of small coal are much more liable to spontaneous combustion than large sized. Bouwman + Freiks(10) confirmed this, but only to a size above 0.1mm, below which further reduction in size had no effect on liability. They suggested that this was due to the effects of macropore and micropore oxidation. Below this critical size, oxygen can fully penetrate a particle of coal. Above it, oxygen can only reach the surface regions and areas accessible through interconnected pores in the coal.

Pores in coal were originally discovered by van Crevelen + Zweitering(11), and their existance was confirmed by Gan et al(12). Gan found that pores varied in size from macro to micro. Low rank coals were found to have porosity arising mainly from macropores, which effectively subdivide the coal particles, and have a high permeability to most gases. Initial oxidation is believed to be a macropore process, and as the temperature rises this progresses to a

micropore process.

Smith(13) suggests that oxygen can diffuse into low rank coals, and that oxidation can take place over a larger surface area initially. High rank coals are not penetrated to any degree by oxygen, so only the outer surface reacts, and becomes inert. Schmit + Elder (9) found that by eliminating the fraction sized below 0.5mm, the heating effect in coal piles was greatly reduced.

2.4 MEASUREMENT OF OXIDATION CHARACTERISTICS OF COALS TO ASSESS LIABILITY TO SPONTANEOUS COMBUSTION

Coals with a high capacity to absorb oxygen, and an accelerating rate of oxidation as the temperature rises, are more liable to heatings than those with a low capacity, and little change in oxidation rate (14).

Several different methods have been devised to measure the oxidation characteristics of coals. All methods cannot be directly compared, as each only produces a relative guide to liability. Results of different workers using the same method cannot be compared unless the method is standardised. What is done is to compare the results from different coal samples, and assess each for its relative liability.

The methods of assessing liability are all based on the oxidation characteristics of the coal, and can be divided into three groups:

1. Measurement of oxygen absorbtion

2. Chemical methods

3. Measurement of the thermal effects of oxidation

2.4.1 MEASUREMENT OF OXYGEN ABSORBTION BY COAL

There are three methods of measuring oxygen absorbtion, manometrically, by gas analysis and volumetrically. In all methods the temperature is kept constant for the duration of the test.

In manometric testing the sample of coal is placed in an enclosed chamber, and the fall in gas pressure during the test is measured. The oxygen absorbed can be determined from this.

Volumetric testing is similar in that the coal is enclosed in a series of jars. At intervals jars are removed, and the gas in them analysed to determine the oxygen absorbed.

The gas analysis method has air drawn steadily through the sample of coal and, the exit gas is analysed.

By altering the temperature at which the test is carried out, characteristic curves can be constructed for each coal, plotting oxygen absorbtion against temperature.

Experiments by Winmill (4) led him to three conclusions. Coals with an absorbtion capacity of 300cm^3 of oxygen per ounce of coal in the first 96 hours are liable to heatings. The greater

the initial rate of oxygen absorbtion, the higher the rate of heat output, thus increasing the temperature and oxidation rate. Low rank coals increase in oxidation capacity with a rise in temperature, but high rank coals remain relatively unaffected by temperature changes.

2.4.2 CHEMICAL METHODS

Chemical methods of assessing the oxidisability of a coal were in favour at one time with scientists in the eastern bloc countries. The methods have fallen out of use because the reactions with coal do not occur in pratice. The results of such tests do not truly represent the way a coal oxidises.

The basic method was to treat a coal sample with an oxidising chemical, such as hydrogen peroxide. A much faster oxidation reaction takes place. The susceptibility of a coal to oxidise can be assessed from the carbon dioxide produced, the amount of chemical used in a complete reaction or the temperature reached in the reaction (7).

2.4.3 MEASUREMENT OF THE THERMAL EFFECTS OF OXIDATION.

Several different methods have been developed for measuring thermal effects. Until each method is standardised it will not be possible to compare results that are obtained by different workers.

In the crossing point method, a sample of coal is placed in an oil bath. The bath is heated at a constant rate, and the temperature of the bath and coal recorded. The temperature at which the coal becomes hotter than the bath is the crossing point or ignition temperature. The lower the crossing point temperature, the more liable the coal is to spontaneous combustion. Figure 2.1 is a typical trace.

A similar method is the differential thermal analysis. A sample of coal and a sample of an inert reference material are heated at the same rate. The temperature of each sample is recorded, and the temperature difference plotted against the temperature of the inert reference. This produces characteristic shapes for liable and non-liable coals, Figure 2.2 (15).

The above methods involve heating the coal at a steady rate, and examining the reaction. This occurs over a relatively narrow temperature range. In order to examine the oxidation characteristics of coal over a wide temperature range, the derivative thermogravimetric analysis method is used. A derivate thermogravimetric analyser monitors the rate of weight loss, or gain, of the coal sample as a function of temperature. The results are plotted, and characteristic shapes are obtained for low, medium and high rank coals, Figure 2.3a, 2.3b, 2.3c. (13)

In order to more closely simulate the conditions that prevail underground, tests need to be performed under adiabatic conditions, where there is no external energy input of loss from the coal sample. Different workers have all developed different

apparatus to make calorimetric measurements. The basic method is the same in each, a typical apparatus is shown in Figure 2.4, A coal sample is held is an oil bath. Gas is passed through the sample, and as the temperature of the coal increases, the temperature of the bath is raised. This ensures that there is no heat loss or gain to the coal, and that any temperature rise is due to the coal oxidising. The temperature of the coal is plotted against time, Figure 2.5 shows a set of curves obtained.

The greater the temperature rise, the more liable the coal is to spontaneous combustion. This method has served to illustrate the role that atmospheric water plays in spontaneous heatings.(16)

The work by Demirbilek led to the postulation of a risk index for a coal. Given the composition of a coal, the underground and atmospheric conditions prevailing, a risk value for the coal can be calculated.

FIGURE 2.1 TYPICAL CROSSING POINT TEMPERATURE CURVE

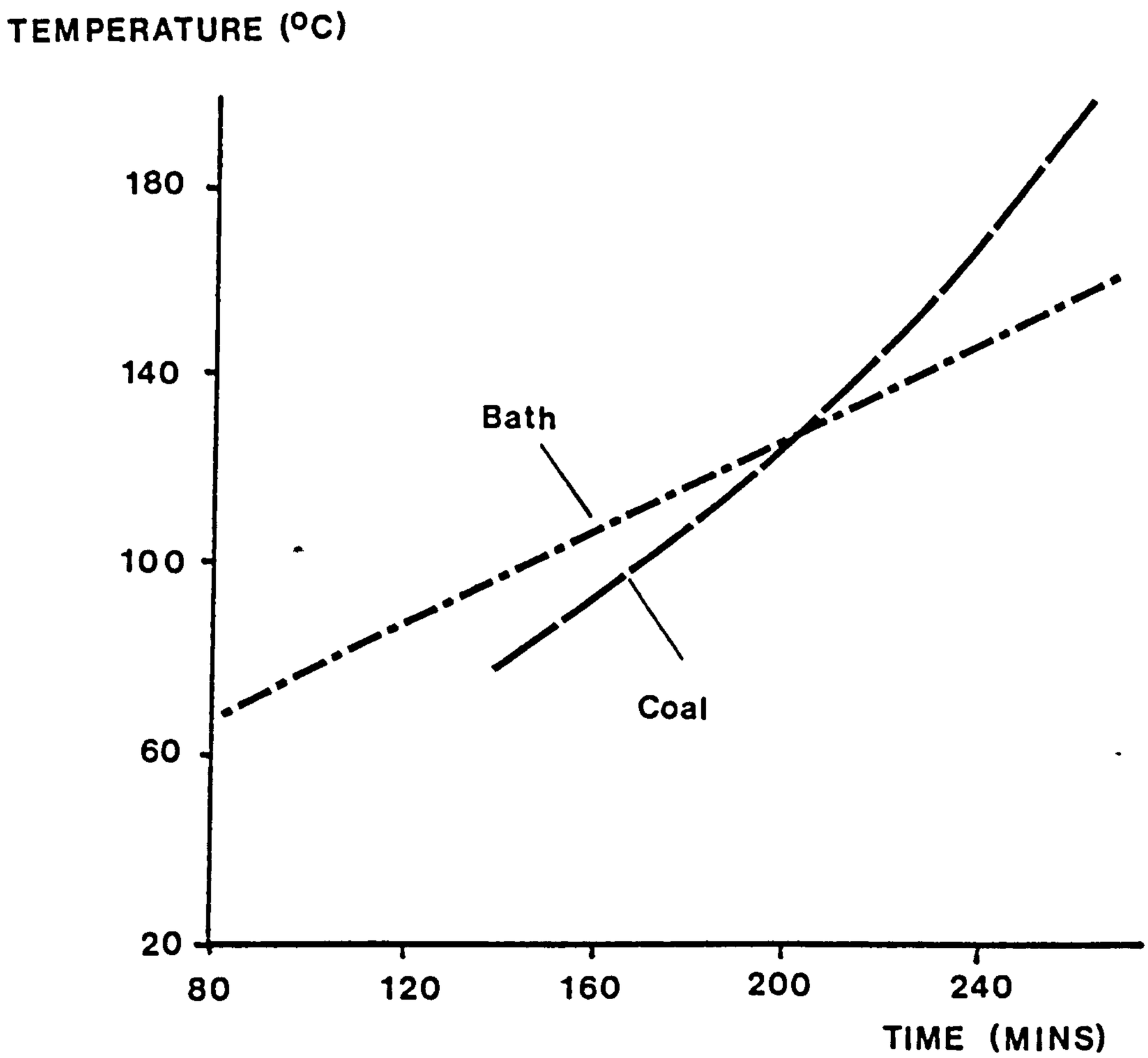


FIGURE 2.2 THERMOGRAMS OF LIABLE AND NON-LIABLE COALS

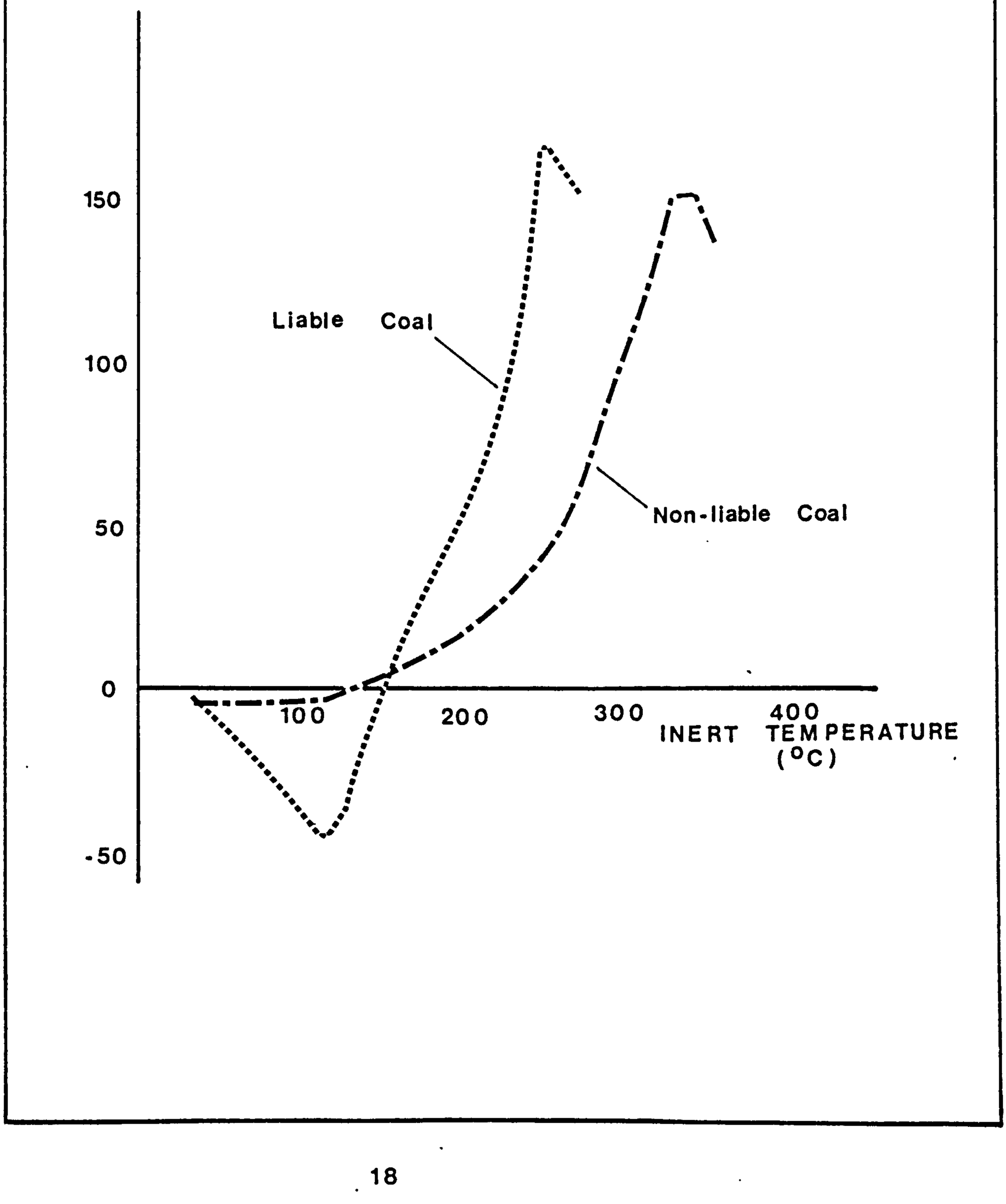


FIGURE 2.3 D.T.G.A CURVES

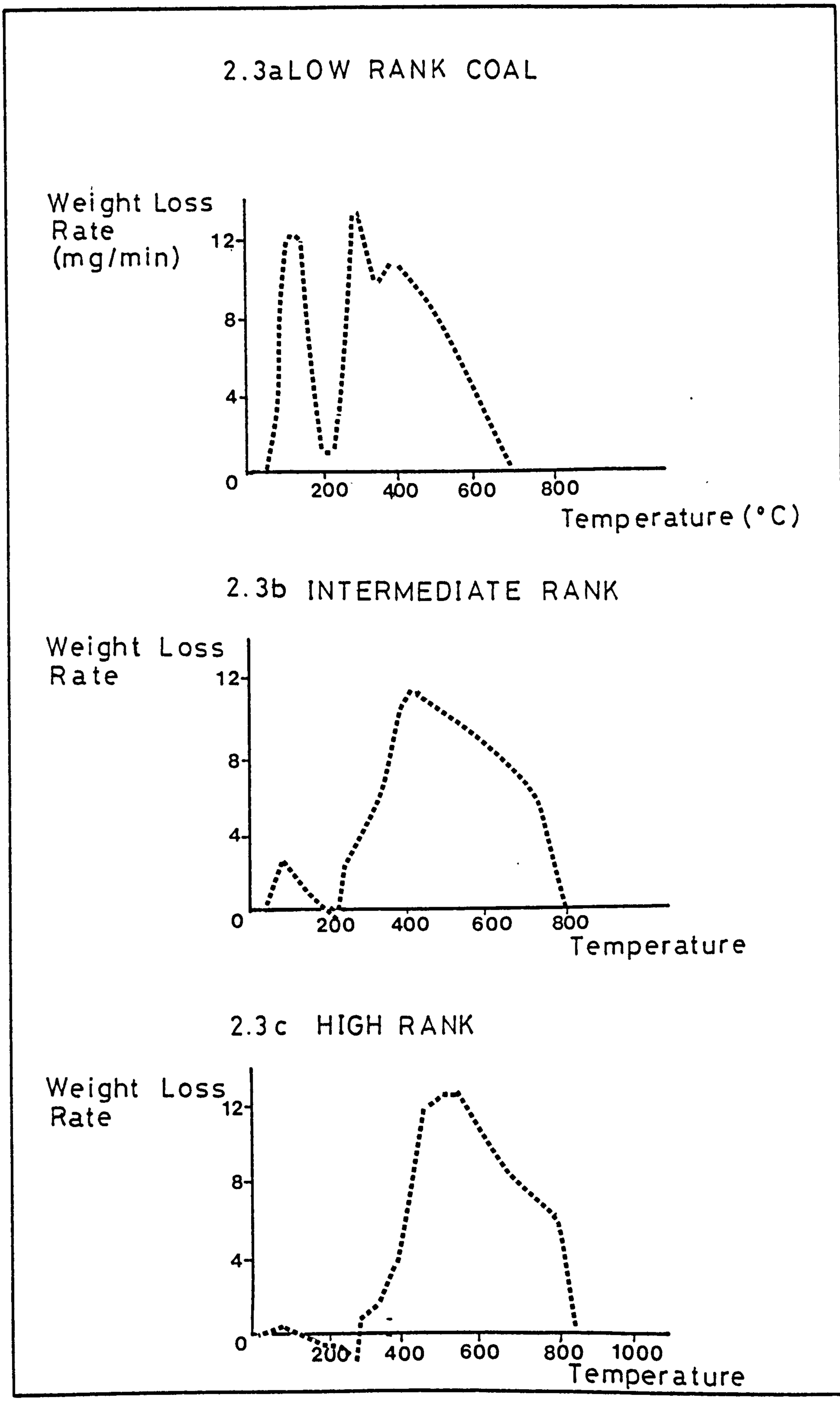
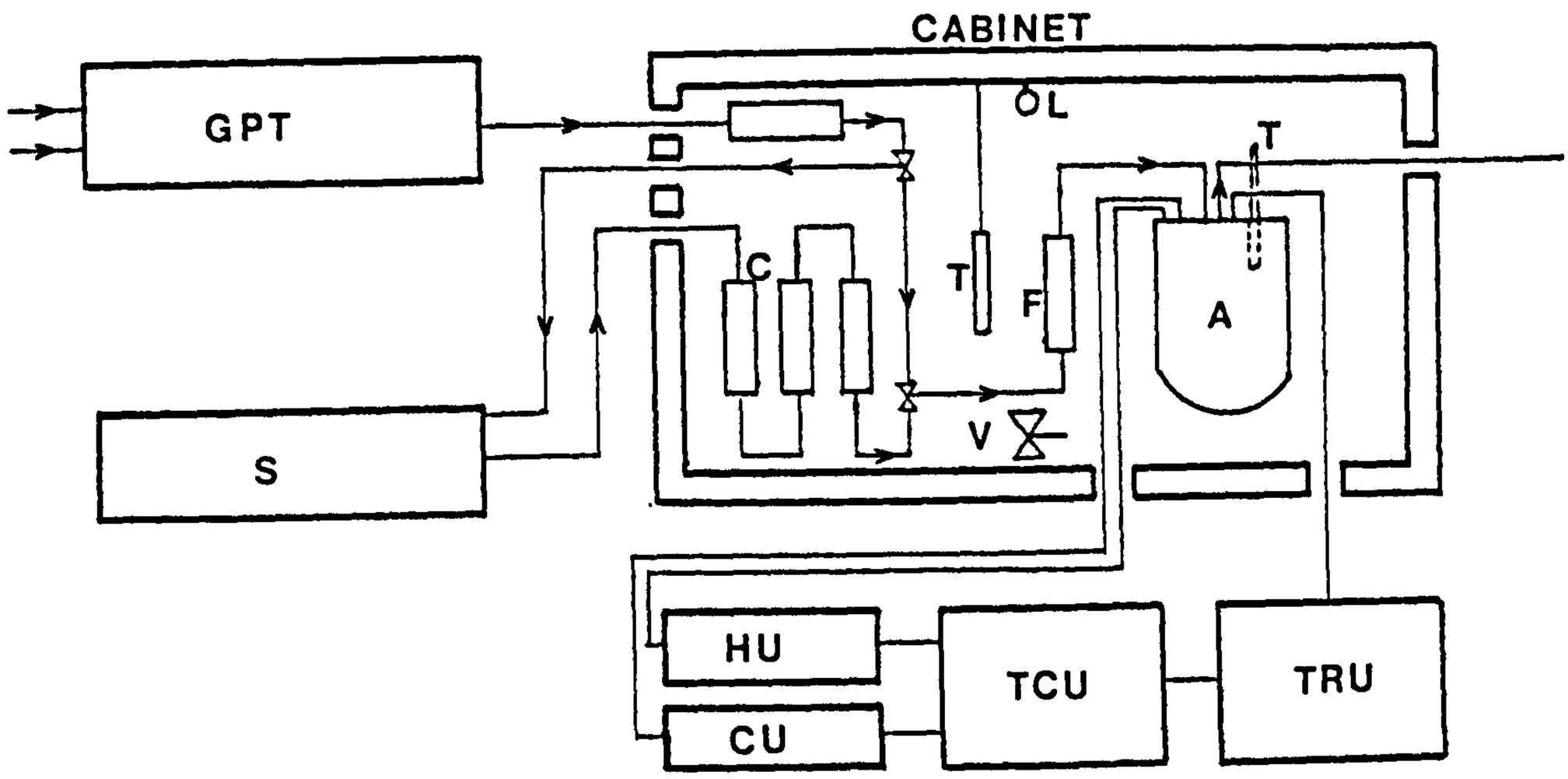
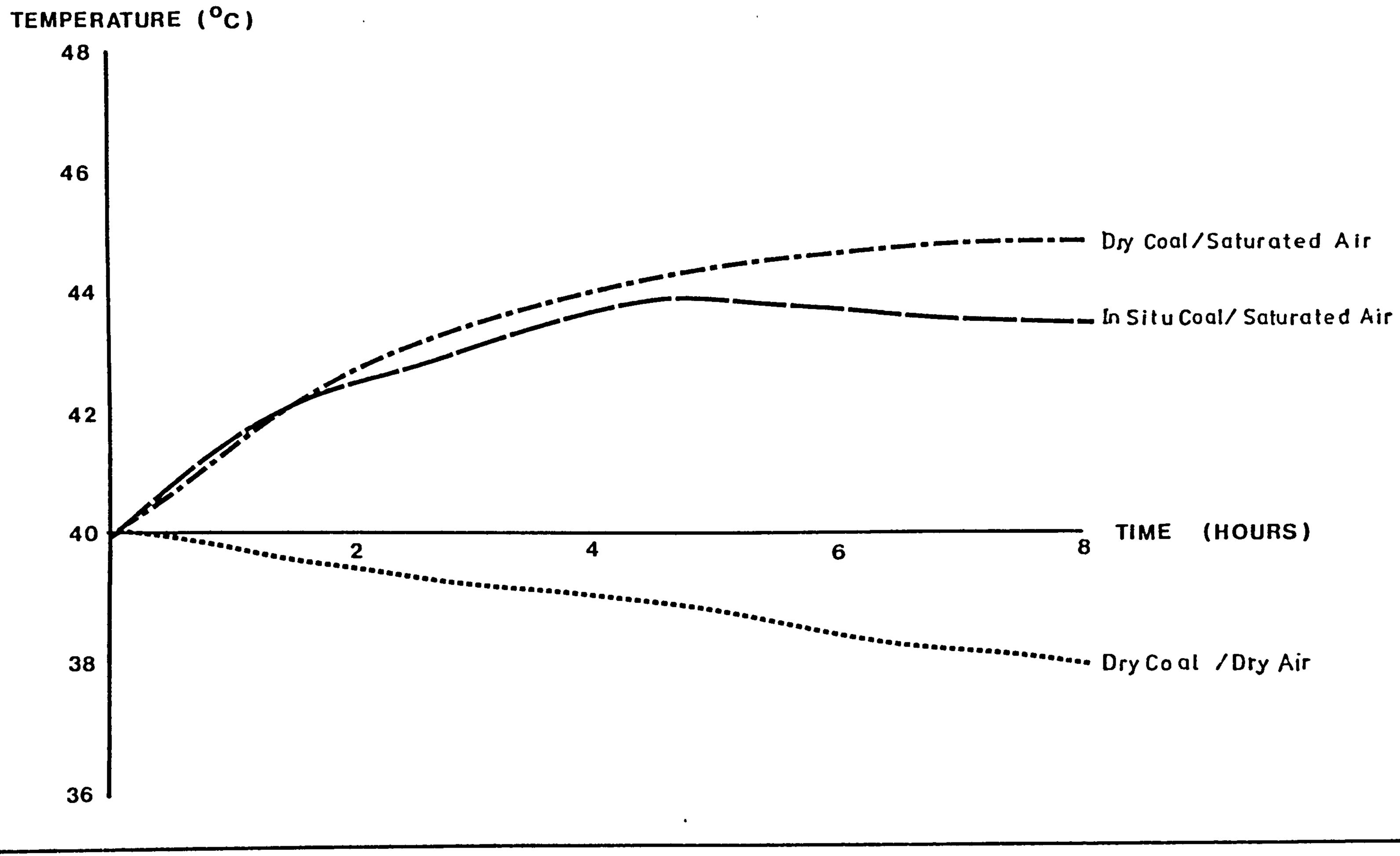


FIGURE 2.4 ADIABATIC CALORIMETRIC APPARATUS



A Adiabatic Apparatus
C Copper Coils
CU Cooling Unit
F Flowmeter
GPT Gas Purification Train
HU Heating Unit
L Lamp
S Saturator
T Thermometer
TCU Temperature Control Unit
TRU Temperature Reading Unit
V Ventilator

Figure 2.5 Adiabatic Reactivity Curves (16)



2.5 GEOLOGICAL FACTORS

As mentioned earlier, the depth of workings has a great influence on the liability of a seam to spontaneous combustion. Other geological factors are the thickness of the seam and the properties of the surrounding strata.

That thick seams of coal are more liable to spontaneous heating is shown by the evidence over the 10 year period 1969 - 79, when half the incidences of spontaneous combustion in the UK occurred in the South Midlands area. This increase in liability is due to the lower rank of thick coal seams, incomplete extraction and possibly that leakage paths across the gob stay open longer because of the thickness of the seam.

Many heatings on longwall faces occur either on a fault line, or when the face advance has been slowed due to bad ground. Usually the critical zone (see later) moves forward with the face. When the face advance is slowed, air may be able to move through the critical zone for long enough to allow a heating to develop in the goaf. Coal may be left in the goaf around a fault, as a result of changing the cutting horizon or a thickening of the seam. Fault planes can act as leakage paths for air through the goaf, and may have fine carbonaceous material in the plane. Bad ground due to cavities and so on is usually packed up on the face with wood chocks, and coal is left in the roof. Again this maintains a leakage path for air, and a supply of fine, oxidisable material. Most wood used underground now is treated with flame retardant

agents.

The fires that take place in colliery spoil tips show that shale will burn spontaneously. The presence of shales with a high carbonaceous content, in the roof or floor, has to be considered a risk. Thin bands of coal and iron pyrites around the seam being worked are additional risk factors. In the goaf, fines are produced from these sources, while in roadways, roof crush and floor lift can fragment the strata and expose it to a steady, slow moving stream of air. A similar effect takes place in gateside packs, where either coal has been used in the pack, or has been left over the pack.

Many coalfields have seams that cannot be worked in isolation. There are usually other seams, often in close proximity to the one being worked. Two cases arise. The first is when the neighbouring seam has been previously worked. The goaf of the other seam is exposed to the pressure difference ventilating the present workings. This becomes a risk if the upper goaf contains quantities of oxidisable material, and the airflow through it is significant. Interaction effects can slow advances, putting both the old and present workings in danger. When an upper seam has not been previously worked, coal may fall into the void created by the goaf below.

The thermal conductivity of the surrounding strata influences the dissipation of heat from the goaf. Shales and coal, which have low thermal conductivities, act as insulating agents to a heating, preventing the heat dissipating, and so aiding incubation.

Sandstones have a higher thermal conductivity, so heat tends to be lost.

In bad ground, leaving a layer of coal on the roof is common practice, to give good roof conditions at the face. In the goaf this coal becomes fragmented, along with the shale roof. Competent sandstone roofs are preferable to weak shales.

2.6 CONCLUSIONS

Coal is a rock containing inorganic and organic components, found in a variety of geological locations. The proportion of one part to another differs from coal to coal. The effect of each of the major constituents of coal on its oxidation potential has been summarised. The effects of breaking coal and the differing geology of coal deposits have on spontaneous combustion liability have been discussed.

The review suggests that the basis for the classification of the spontaneous combustion liability of a coal is based on past experience, laboratory reactivity tests and subjective judgement.

Despite two hundred years of research, no universally accepted quantitative method of estimating the risk of a coal to spontaneous combustion has been found.

The next chapter will discuss the factors that are not a property of the coal, or its location, that affect liability.

CHAPTER 3

EXTRINSIC FACTORS AFFECTING THE LIABILITY TO SPONTANEOUS COMBUSTION

3.1 INTRODUCTION

The mining engineer has no control over the composition or location of the coal he is to mine. He does have control over the method of mining and the underground environment. In order for spontaneous combustion to occur air must be allowed to move through an area of broken coal. The temperature and humidity of the air have an obvious effect on the heating of coal. Preventing such air movement will prevent spontaneous combustion.

In mines liable to spontaneous combustion ideally retreat mining should be used. Where this is not possible and advance mining is practised, low ventilating pressures are used. Bord and pillar mines subject to spontaneous combustion have to leave larger pillars than required for roof control, to prevent pillar cracking

and spalling. It is clear that by suitable choice of mining method, combined with high standards of workmanship, the potential for spontaneous combustion can be reduced.

This chapter considers the effects that environmental conditions underground and the mining method have on liability to spontaneous combustion.

3.2 ATMOSPHERIC FACTORS.

Much research in the past few years has been devoted to investigating the effect that atmospheric factors have on spontaneous combustion liability.

Rises in the temperature, and moisture of the air will tend to increase the risk. The role of changes in barometric pressure is not understood.

3.2.1 TEMPERATURE

Temperature has a great effect on the rate of oxidation, especially on low rank coals. At low temperature a 10 C rise in temperature can double the rate of oxidation (17). In a mine changes in temperature are carried by the ventilating air. The temperature of mine air is largely governed by the temperature at the top of the downcast shaft in shallow mines. Other factors tending to raise the temperature of the ventilating air include autocompression, virgin strata temperature, machinery, strata

movements and some chemical processes.

The rise in air temperature because of autocompression is $1^{o}C$ per 100M of shaft depth. This can be considered an adiabatic process.

The heat transfer from the strata to the air depends not only on properties of the air and the rock, such as the temperature of the air and rock, the moisture content of the air and wetness of the rock, but also on the size, shape and roughness of the roadways, the area of rock exposed and the rate of airflow.

Given steady state air conditions, that is temperature and humidity, the temperature of the exposed rock will be largely dependant on the virgin strata temperature (VST) and the time since the rock was first exposed. The VST is directly related to the geothermal gradient, which is the rate at which the rock temperature increases with depth. This is inversely proportional to the thermal conductivity of the rock.

Rocks which are high in quartz have higher thermal conductivities than those which are low in quartz content, such as shale and coal. In a rock formation composed of layers, such as the coal measures, the thermal conductivity of each layer is different, and so therefore is the geothermal gradient. Carbonaceous strata have a much higher geothermal gradient than quartzitic. The gradient is further affected by geothermal activity, and radioactivity of the rock.

In Britain the geothermal gradient is between 30m/C to 40m/C. Different coalfields have different gradients. Extremes are recorded in Lancashire, 50m/C, and the Beeston seam in Yorkshire at 29m/C.(18)

As mining gets deeper, and the VST rises, the risk of spontaneous combustion increases. This is shown in areas such as the Yorkshire coalfield, where spontaneous combustion was rare when mining was at shallow depths on the west of the coalfield. As the mines have got deeper, and further east, spontaneous combustion outbreaks are more common.

The major man made source of heat is from machinery. All electrical power used underground, unless it is used to do work against gravity, is degraded to heat. With the current "horsepower addiction" being shown by mining management, this heat input from machinery is certain to continue to rise. (19)

The heat input from such sources as chemical reactions and strata movement tends to be negligible.

The three main causes of rise in temperature are autocompression, heat from strata and heat from machinery. The temperature rise can be controlled by increasing the ventilation quantities. This may have a detrimental effect on the liability to heating in longwall wastes. Alternatively air conditioning can be used to cool, and de-humidify, the air in the mine workings. The only control the mining engineer has over the heat input to the air is in the installed power of underground machinery.

The air temperature is one of the most important factors in the rate of oxidation, and the mine temperature has a significant effect on the liability of a seam to spontaneous combustion.

3.2.2 BAROMETRIC PRESSURE

It has been recognised for many years that the rise and fall of barometric pressure has influenced the emission of methane from wastes. Post war work in this field was done by Carter + Durst (20). They further suggested that pressure fluctuations would cause gas movement in and out of sealed areas. This effect was quantified by Jolliffe + Raybould (21) using Boyles law. More recently, Fauconnier + Buekes (22) compared the barometric pressure, the pressure difference across a seal and the analysis of gas samples taken at the seal. This showed that air was moving in and out of the sealed area.

It is likely that fluctuations in barometric pressure will similarly affect the nature of gases in a longwall waste. The rise and fall of atmospheric pressure will cause inert gases to move out of the waste, and be replaced by oxygen. Before scientific investigation began, most heatings were detected on rainy days. Rain is associated with falling atmospheric pressure. It is possible that the increase in humidity may accelerate a heating, but more likely that the fall in atmospheric pressure releases CO into the mine general body air. In addition to this breathing of the waste the changes in pressure may alter the pattern of the migration of air from the intake to the return across the waste.

Barometric pressure changes are of much larger magnitudes than ventilation pressure changes. The meteorological office notifies mines when a drop in atmospheric pressure exceeding 1.3mb per hour is expected. A typical longwall district may be ventilated by a pressure drop of 500 Pa, or 5mb. The atmospheric pressure in the UK can range from 950mb to 1050mb. They may have a much greater influence on the pattern of gas flows, and incidence of spontaneous combustion than is currently thought.

3.2.3 ATMOSPHERIC MOISTURE

As mentioned earlier, low rank coals that have a high moisture content tend to be more susceptible to spontaneous heatings. The part that atmospheric moisture plays in spontaneous combustion liability has been the subject of much debate. There are three conditions that need to be considered:

1. When the air picks up moisture from the coal
2. When no exchange takes place
3. When the air loses moisture to the coal

It is known that workings that are wet, or have been flooded, are more liable to heating than others. Fauconnier + Buekes (23) note that mine fires in South Africa start around 2 months after the onset of the dry winter conditions, after the humid summer. The value of their work is open to discussion. The humidity of the atmosphere was obtained from a surface weather

station some 80km from the mine, and no long term barometric measurements were taken. Futhermore, the two fires that the work is based on were both detected at advanced stages. The conclusion that they reach is that an equilibrium is reached between the coal and the air during the summer months. The drop in humidity in winter will disturb this equilibrium. According to the work of Guney (24) this will cause liable coals to heat spontaneously.

This would agree with the findings of some workers, that water removes oxidized products from the coal surface, and decrepitation occurs, exposing fresh surfaces for oxidation. Moisture that evaporates off the coal, while resulting in a loss of heat, will recondense nearby and act as an insulating agent for the heating. Due to the latent heat of condensation, the temperature of coal that water condenses on is raised. The other side of this coin is that coal which has had water condense on it may be less liable as the result of the addition of the heat capacity of the water, the latent heat of vapourisation, and a thin film of water covering the active surfaces of the coal preventing oxidation.

Both Richters (3) and Winmill (4) pointed to the effect of moisture on the oxidation rate of iron pyrites in coal. Winmill did not consider pyrites to be significant. Richters stated that when a large proportion of pyrites was present the rate of oxidation was increased with a rise in atmospheric moisture, otherwise it was reduced.

Extensive work was done in the 1960's by Guney, Hodges + Hinsley (25), and by Hodges, Hinsley + Bhattacharyya (26). They examined the three conditions, mentioned earlier, in adiabatic experiments. They concluded that the condensation of water onto coal raises the temperature significantly, and that moisture is responsible for the spontaneous heating tendency at low temperatures. This is due to two factors. The latent heat of the water as it condenses on the coal, and the heat of hydration effect from the hygroscopic nature of coal. When no exchange of moisture took place between the coal and the air, there was no change in heat production. When the coal lost water to the air, it was stated that the risk of heating may be reduced.

3.3 MINING PARAMETERS

The control of the mining operation is one of the few ways in which the mining engineer can influence the occurrence of spontaneous combustion. In seams that are known to be liable, the risk of heatings is a major factor in the design and planning of operations.

The are three areas to be considered, main roadways, gate roadways and longwall faces

Bord and pillar workings do not fall into the area of this thesis. In areas that are liable to spontaneous combustion that are mined using this method, large pillars are left to prevent the coal

cracking, and exposing surfaces that can be oxidised. Piles of fine coal are cleared away from the roadways, and the worked out areas are sealed off well.

3.3.1 MAIN ROADWAYS

The chief hazard in main roadways lies in the slow passage of air through crushed and fractured coal. Sites where this happens are air crossings, junctions, air doors and regulators. Coal in the roadway sides and floor is prone to crushing and breaking.

There can be high pressure differences between the intake and the return, which may force air to travel through the intervening crushed strata at air crossings, doors and regulators. Such a situation arose in an air crossing at Kellingly (27). The wedges of coal were crushed, and the roadway had not been sealed. The stonedust in the roadway acted as an insulating agent. Because of the high airflow, the heating was not detected until an official noticed the smell. By this time the heating was well established, and had to be dug out. At no time were high concentrations of CO detected.

At corners and junctions, especially those with acute angles, the coal is crushed. The air follows the most direct route, through breaks in the coal, over the fines caused by crushing. There is an obvious risk of heating.

Fractured coal around air doors and regulators provides a path for air to travel through. Higher pressure differences lead to higher airflows through the coal, thus doors between a main intake and return are most at risk. A similar situation arises when there is resistance to airflow in the roadway, such as mine cars and stacks of material. The extra resistance can cause the air to travel through fractures around the roadway. Such resistances should not be left in one place for long enough to cause a risk.

The practice of using main roadways as development gates is a poor one. A fire on the district may then cause serious problems for other workings in the mine, and put a block on future developments.

Practical prevention of spontaneous combustion in main roadways can be simple. Air doors and regulators must be sited so that the movement of men does not cause short circuits, and should be keyed into the roadway sides during construction, to prevent air moving through broken coal around the roadway. Sharp angles at roadway corners can be avoided. In thick seams roadway stability can be improved by driving the roadways out of the coal, also having the effect of reducing spontaneous combustion liability. Other precautions (28) include injecting bentonite mud, or grouting ground that has become broken, especially around air doors. Roadway linings can be sealed with cement or gypsum products.

3.3.2 GATE ROADWAYS

Gate roadways suffer from similar problems to main roadways, to which the solutions are the same.

When planning face layout consideration must be given to the need to seal off districts in isolation. A fire at Fryston (29) resulted in the loss of two faces and a considerable area of reserves. The fire occurred in an area of total extraction, where it proved impossible to control the leakage through the wastes. As a last resort seals were put on the main roadways leading to the area.

Deep seated goaf fires are caused by the slow movement of air from the intake to return across the goaf. This air movement can be reduced by keeping the pressure difference across the goaf to a minimum, either by the use of pressure chambers or by keeping ventilation pressures low. Gate roadways need to be large enough to carry the ventilating air without excess resistance. Gates should be kept clear of unnecessary obstructions.

3.3.3 LONGWALL COAL FACES

Retreating and advancing faces present dissimilar problems. Advance faces are considered to be at greater risk due to the air leakage across the goaf.

The rate of face advance is critical. The reloading of the caved waste consolidates the goaf and the gate side packs, effectively sealing them and preventing airflow. The sooner this happens, the less likely an area is to be subject to a critical airflow.

As mentioned earlier, all coals oxidise to some degree on exposure to air. Given a coal that is liable to heat, the airflow in the waste can affect the occurrence of a heating. If the flow is too slow, insufficient oxygen is supplied to sustain combustion. If the flow is too fast, the coal loses heat to the air, and combustion does not take place. In the zone of critical flow, the oxygen supply is sufficient to sustain combustion, and the heat loss is small enough not to prevent it. The zone of potentially critical airflow is close behind the face (30). On a retreat face, the leakage through the goaf is much smaller, and confined to a zone just behind the face line, so the rate of advance is less important.

The time a zone is exposed to critical airflow is important. The air supply has to be sufficient for oxidation to occur, but insufficient for heat generated to be carried away. These conditions must persist for long enough for the oxidation rate and heat to build up to cause a self sustaining heating. These conditions are likely to persist when normal face advance is interrupted, or there is a leakage path across the goaf. Reasons for the face advance being interrupted are well known, but include industrial disputes, salvage and geological problems.

In regions that are liable to heatings, routine preventative methods are laid down in the area Code of Practice (31).

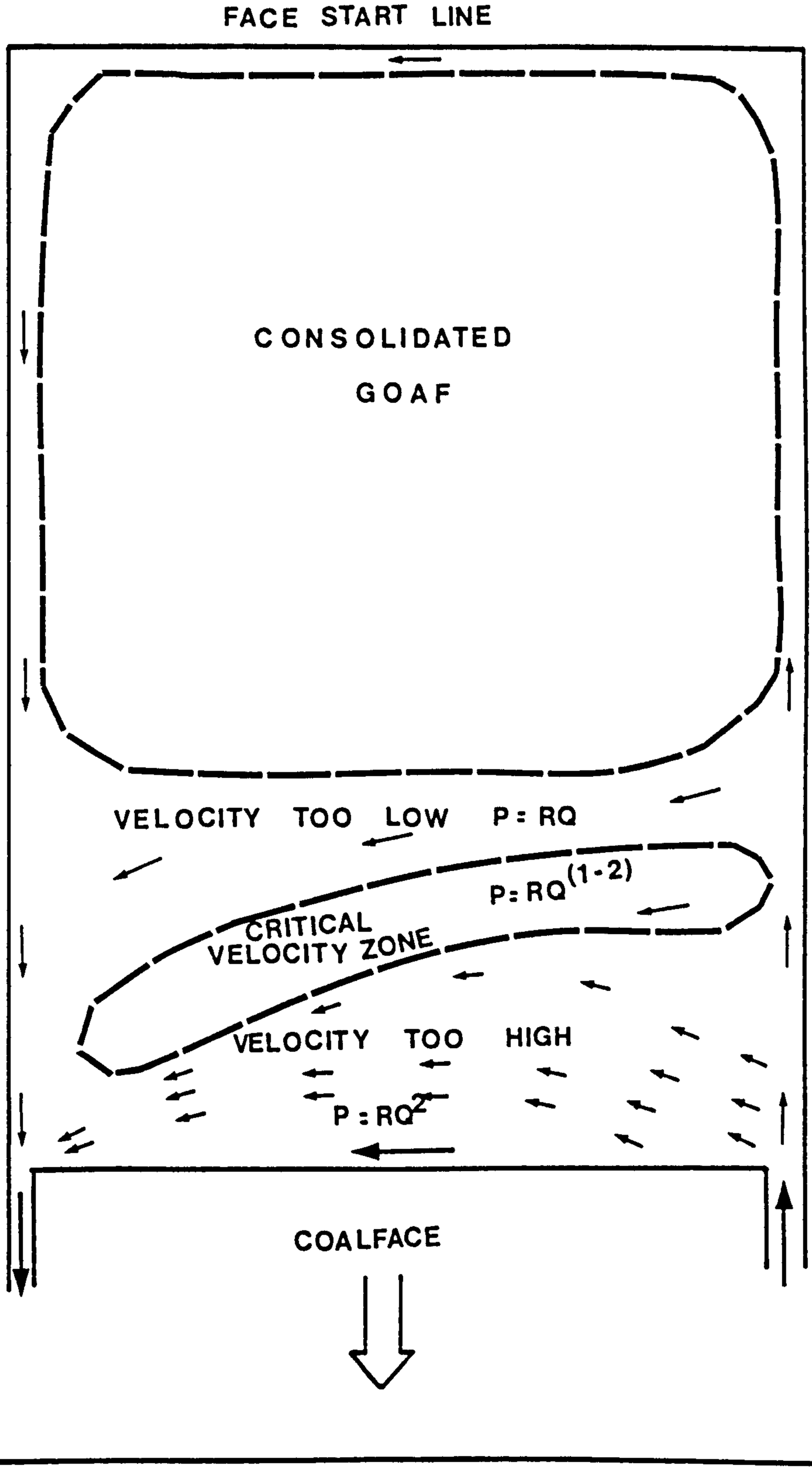
3.3.3.1 Retreat Faces

Retreat faces are the standard method of working in many high risk areas, as the risk of heatings on retreat faces is much less than that on advancing.

It would appear that retreat faces in multi-lift panels are at greater risk than those in single lifts, this has been mentioned earlier. Lord (30) details two heatings on retreat panels in the lower leaf. Sealed off old workings are usually at a different pressure to the present workings. This will cause a slow air movement through the goafs.

The airflow pattern proposed by Lord for a retreat face is shown, Figure 3.1

FIG. 3.1 AIRFLOW DISTRIBUTION ON A RETREAT FACE



Air reaches the face with considerable kinetic energy, and penetrates deeply into the waste. Some of this air may travel along the abandoned roadways and pass through the original face start line. The start line is not considered to be at risk once normal advance is achieved as the air leakage is believed to be small, and any heating that develops will be quickly buried. Flow through the consolidated area of the waste is minimal. The main flow will occur at the front of the waste, and this will return to the face airstream towards the return gate.

Because retreat faces move quickly, the critical zone is not usually subjected to airflow for long enough for a heating to be incubated. Problems may occur at the start of production, when the waste has not been consolidated. Air may then be free to travel through the waste, or into old workings above. A greater risk is the face stop line, while the equipment is being salvaged. The unconsolidated waste may be scoured by air for several weeks.

These problems can be reduced by stopping the abandoned roadways as the face retreats. When the face stops the ventilating quantity of air may be reduced. In situations where it is intended to re-use a gate roadway, monolithic packing should be used so as to seal the waste.

Careful consideration should be given to the site of primary stoppings on retreat panels. The face stop line is usually very close to these sites. If a heating breaks out when the face has finished, it may not be possible to build these stoppings. Secondary stoppings will then need to be constructed. This can

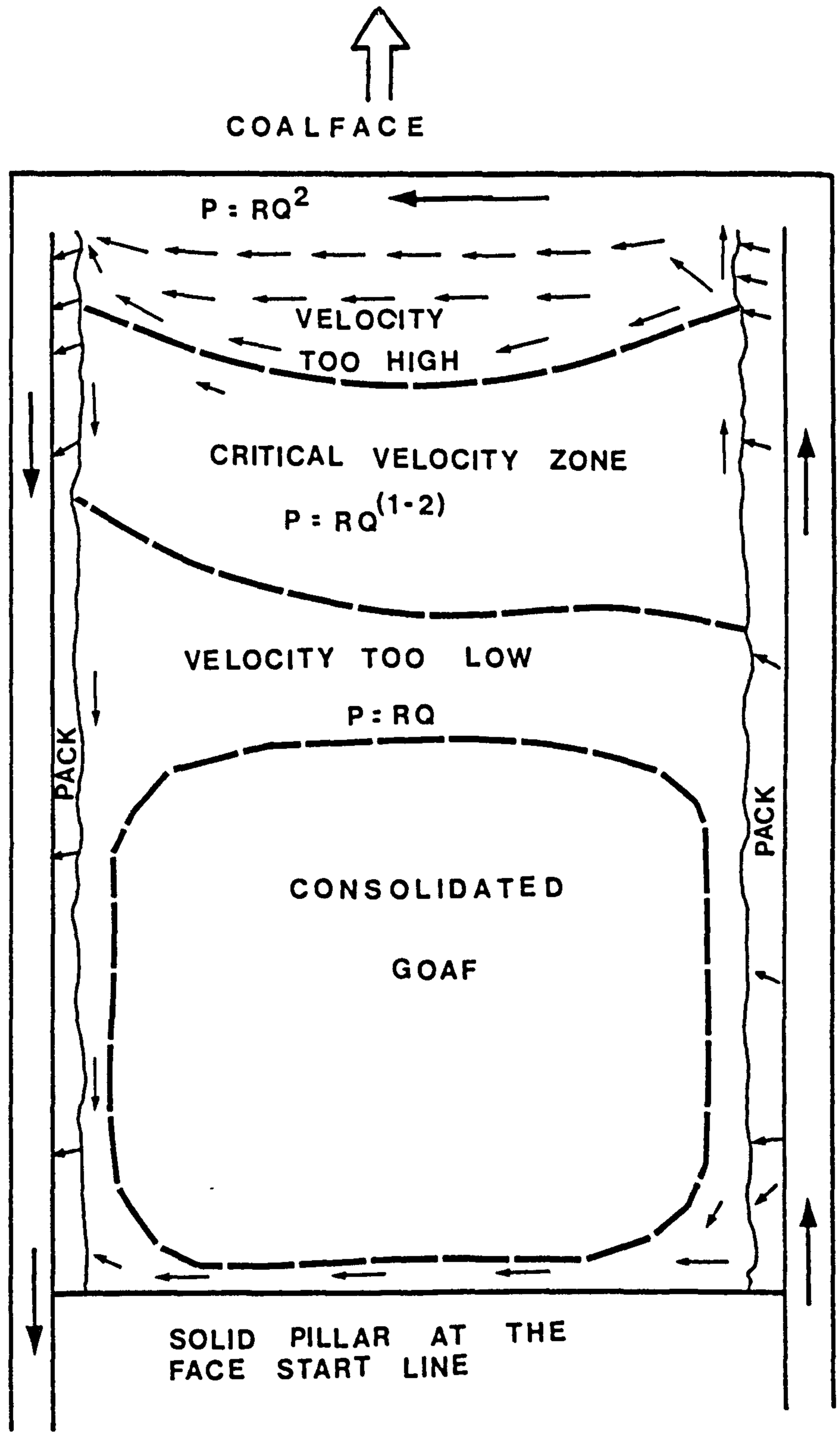
result in a large area of the mine being sealed off. Such a situation happened in March 1981 at Fryston (29). The heating was on 74's, a retreat face which was being salvaged, the equipment being transferred to 76's, a neighbouring advance panel. The primary stoppings proved ineffective, partially due to uncontrollable leakage from the abandoned goafs around. Secondary stoppings were built in the main roadways, sealing off a large area of future development. Using nitrogen from a surface borehole, the heating was brought under control, and the primary stoppings were subsequently adequately built. The cost of the operation was some £300,000 to save equipment valued at £3.5million.

To prevent a recurrence of this, explosion proof primary stoppings have been developed. These are built into a roadway as the face is installed, and can be closed and sealed quickly when needed.

3.3.3.2 Advance Faces

The airflow zones suggested by Lord on an advance face are shown, Figure 3.2. Work by Highton (32) suggests that the air tends to move straight across the goaf. There are three areas at risk on the advance face, the face start line, the packs and the critical velocity zone in the waste. The object in all three areas is to prevent the through movement of air.

FIG. 3.2 AIRFLOW DISTRIBUTION ON AN ADVANCE FACE



The face start line is the area most at risk. The solid pillar provides effective support, preventing consolidation of the waste and packs. The full pressure drop around the district is applied across the start line. In thick seams the waste is likely to be full of fragmented coal, while crush on the pillar edge will generate more fine coal. There is thus a leakage path, for the duration of the life of the district, through fine coal, across which the full ventilating pressure on the district is applied. Efforts must be made to prevent airflow through the face start line. To promote consolidation, supports for the original face drivage must be removed, along with all wood props at the start line. Packs are often keyed into the solid coal pillar, and sealed using injection techniques or built using monolithic packing systems. The waste strata can be injected with inert material, such as bentonite, that acts as a sealant. The roadways either side of the start line may be sealed. This is usually done for between 10 and 30m either side using shuttering and plaster. In some cases, more often where a heating has been detected on the face start line, pressure chambers are built to prevent the flow of air. As the face advances further, the pressure difference may have to be raised to maintain ventilating quantities at the face. Work which was satisfactory at the start of the face life can then prove inadequate to prevent leakage.

The gateside pack has two functions to perform. The first is to prevent convergence of the roadway, and the second is to prevent air leakage across the goaf. The packing method selected is dependent on the supply and cost of materials brought into the mine,

as compared with the availibility of ripping dirt, and the need to prevent roadway closure. The pack should have a uniform strength, resist air leakage and be constructed from inert materials. (33)

In many collieries packs are handbuilt from ripping debris, which contains carbonaceous material, and coal. This represents the cheapest method of pack construction. In the UK it is common practice to leave a coal roof to ensure good roof conditions in the packhole. An accumulation of unconsolidated, fine coal builds up in the packs, while overbreak from the roof provides leakage paths. Leakage paths through packs exist, for air moving from intake to return across the goaf and for slow moving air down the roadway. The advance of the face, and subsequent reloading of the waste, closes and tightens packs. Ribside packs are not subjected to the same loading as goaf side packs, and are much more porous, resulting in favourable conditions for heatings. In seams liable to heatings, ribside packs are either avoided, or sealed to prevent the flow of air.

Leaving timber supports in packs must be avoided. The resistance of the timber prevents consolidation around the support, and causes cavities and fine coal. The leakage path that exists can result in a heating. Timber is not the initiating cause of pack heatings, as used to be thought. Untreated wood has a lower ignition temperature than that of coal, and so may help to sustain a heating. With good workmanship and construction, conventional packs can be adequate. Sheeting and cross walls in the pack can be used to prevent leakage. The roof can be prevented from breaking up by

giving support while the pack is built, and ensuring that the pack is made tight to the roof.

In some situations it may not be feasible to use ripping dirt, in thick seams or where in line face ripping is practised. In these cases packs are built of coal or material imported to the mine. Using coal, which is either mechanically stowed or in bags, to make packs has obvious heating risks. Imported materials must be cheap and easy to transport to be considered.

The original external material used to build packs was wood. Wood chock packs have good resistance to closure, but do not seal the waste in any way. Unless treated wood will ignite at a lower temperature than coal, and so propagate a fire. Over the years timber has become an expensive alternative.

Lightweight aggregrate blocks do not have as good an overall strength as wood. In the initial stages of loading the blocks are strong, but offer little resistance after failure. By constructing packs solely from agalite blocks, a pack which is initially strong, inert and seals the roadways well can be obtained. By using block walls with a dirt filling, and possibly a sealant, a strong pack which is impermeable and relatively inert can be built. A similar effect is obtained using wood and dirt. Using wood and agalite blocks requires the transport and storage underground of large quantities of bulky materials.

Monolithic pump packing is frequently used in thick seams and those regarded as high risk (34). There are many different proprietry brands of pump packing materials, but only two basic methods. One uses two pumping stations. Sodium carbonate and bentonite, and cementaceous materials are separately mixed with water, and pumped into the pack hole. The alternative is to mix the material at the pumping station. The fill is pumped in one pipe to the pack, where it sets in the pack bag. This is simpler to use, and results in less material being wasted, as there are fewer partly used supplies of material. The drawback is that the pipe range has to be flushed through after use, before the fill sets in it. The packs swell tight to the roof and floor. If there are no breaks in the pack, an impervious seal is formed. To contain the fill while it sets, it can either be pumped into a pre-formed bag, or contained by shuttering on three sides and the wall of the last pack on the fourth.

The packs are quickly constructed, setting hard in 30 minutes, and progressively hardening to bear load. Although originally not as strong as agalite blocks, pump packs do not crush and fracture. The materials used are inert and non-flammable, although coal may be used to give bulk to the mixture, and some of the constituents, such as lime, are mildly corrosive.

Transport of the materials required for pump packs is easier than agalite and wood blocks. Less solid material has to be moved, and the pumping/mixing station can be remote from the face

ends, with a pipe range to transport the mix to the face. Improved materials are becoming availible. These shrink less, set faster and are cheaper than previously availible materials.

In areas of British Coal where faces are regarded as high risk, or roadways are to be re-used, guidelines are specified for packing methods. Often monolithic packing is the required method.

The third area at risk on an advance face is the critical velocity zone in the waste. At normal face advance rates this zone is continually moving forward, so it is not given time to incubate a heating. Problems may occur when a face is delayed, or stopped for salvage. Similarly fault lines through this zone can keep open an airflow path. The condition is similar to that at the face start line. The packs and goaf do not consolidate because of the support provided by the coal pillar. The fine coal in the goaf is exposed to a steady airflow. The basic precaution is to prevent the flow of air through this zone. This is achieved by ensuring that as the face reaches the end of its life, the packs are built to a high standard. The roadway sides may be sealed by shuttering and plaster, bentonite mud or grout can be pumped into the strata. In addition to these steps, nitrogen has been used on some salvage faces when it is anticipated that the length of salvage will exceed the incubation time for the coal. The initial results of such use have been encouraging. Areas of British Coal which suffer from spontaneous combustion outbreaks lay down maximum times that are allowed for the salvage of face equipment, for example 8 weeks in South Derbyshire. (27)

3.3.4 CONCLUSION

This chapter has discussed the direct effect that air temperature and humidity have on the spontaneous combustion liability of a coal.

The main situations in which a longwall operation is liable to spontaneous combustion have been identified. It has been shown that the risk, and losses from, spontaneous combustion can be reduced substantially by a suitable layout of the mine and choice of mining method.

It is clear that precautions against spontaneous combustion are vital in areas at risk. In the event of a heating, early detection and combat are needed to increase the chances of successful counter measures. These aspects are examined in the next chapter.

CHAPTER 4

THE PREVENTION, DETECTION AND COMBAT OF SPONTANEOUS COMBUSTION

4.1 INTRODUCTION

Early and accurate detection of a spontaneous combustion heating is vital if the outbreak is to be successfully controlled. In the past heatings were often not detected until an advanced stage, but modern equipment has resulted in much earlier detection.

Prevention and combat of spontaneous combustion are two sides of the same coin. The same techniques are often used for both prevention and combat. In most methods the aim is to exclude oxygen from the site of a potential or active heating.

This chapter considers the methods that are used to prevent, and if this fails, detect and combat spontaneous combustion.

4.2 THE DETECTION OF SPONTANEOUS HEATINGS

There are two main methods of detecting heatings, physical signs and air sampling.

4.2.1 PHYSICAL SIGNS

When a coal oxidises, and the temperature rises, the volatiles are evolved. These have a characteristic smell, which varies for different types of coal. As the heating progresses the nature and intensity of the smell changes. Initially the smell is faint, and may be difficult to distinguish from the normal underground odours.

As the temperature of the coal rises, moisture and other light volatiles are driven off. When these come into contact with cooler surfaces they will recondense. This gives rise to the sweating of roadway supports. Care needs to be taken not to confuse this with normal mine water, or the condensation of normally saturated mine air. Similarly, when the moist, warm air from a heating meets the cooler mine air, the water will recondense. This produces a cloud of visible water vapour in the air, sometimes mistaken for smoke.

Heatings close to the workplace can sometimes be noticed by hot spots on the roadway sides, or warm air which has passed over the heating. Sometimes the rise in temperature on the roadway sides can be felt with the hands. A more sensitive method of detection of

roadway hot spots using infra-red viewers and sensors has recently been evaluated in the UK (35). Infra-red viewers are much more sensitive, and can be used to detect warm spots at a very early stage of the heating.

On occasions a heating may progress so quickly that the first indication is the emission of smoke, or flames into the roadway. Such fires are usually very close to the roadway sides.

4.2.2 VENTILATION SAMPLES

The main products of combustion underground are carbon monoxide, carbon dioxide, methane, water and other volatiles. Carbon monoxide is the most valuable in the detection of oxidation and assessing the progress of the heating. The production of oxides of carbon results in a loss of oxygen from the atmosphere. The level of carbon monoxide in the ventilating air is also affected by blasting fumes, diesel exhausts and changes in atmospheric pressure. Fog on the surface alters the CO content of the air entering the mine. Care must be taken to distinguish between these causes of CO rise and a fire or heating.

The underground atmosphere is always deficient in oxygen when compared to fresh air. The proportion of this deficiency due to oxidation reactions has to be calculated. To calculate the percentage of oxygen absorbed by oxidation it is necessary to know the nitrogen content of the air. From this the percentage oxygen that should be present to correspond with the percentage nitrogen

can be found. The oxygen difference due to oxidation is found. The Graham ratio is:

(CO% / Oxygen difference) * 100%

This ratio rises as heating activity becomes more intense. Due to the ever present oxidation underground, each face has a characteristic value. This varies from face to face. A rise above the normal ratio on a face often indicates a heating.

In order to calculate the Graham ratio a complete gas analysis is needed. These are taken using a hand pump and some form of gas container to collect the gas, while the analysis is done at a local laboratory. General body samples are taken and analysed at least weekly under normal working conditions.

In order to obtain a quick assessment underground of carbon monoxide levels, hand sampling devices are used. These only detect CO, and are prone to erroneous results by being affected by diesel and blasting fumes. Hand held, electronic instruments are useful to give quick readings. They can be used to give spot readings, like the Pitco, or can continuously monitor through a shift, such as the North Western CO detector. This will sound an alarm if the CO level rises above a pre-determined setting.

Several different types of chemical means of detecting CO are availible. Most rely on the colour change that palladium sulphate undergoes when in contact with CO. The most common in UK mines is the Draeger tube. This consists of an aspirator bulb and glass tube. The ends of the tube are broken off, and the

appropriate end inserted into the aspirator. The chemical in the tube discolours brown from the inlet end, the length of discolouration being proportional to the CO content and number of aspirations. Tubes with different CO ranges are available to measure from 10 to 3000 ppm.

All general body air sampling methods can be influenced by carbon monoxide generated from other sources than coal oxidation. In the Western Area pipes are left over the pack every 50m to inject sealants into the waste (32). Weekly gas samples are taken through the pipes. By doing this, oxidation in the waste is monitored. Heatings can be detected at a very early stage, often before the general body samples show any rise in carbon monoxide content. Care needs to be exercised in the assessment of results from sample pipes. Different seams appear to have different waste oxidation characteristics, and the normal rise in carbon monoxide in the waste cannot be taken as indicating a heating.

4.2.3 TUBE BUNDLES

The best way detect a heating is to monitor the mine atmosphere continually. In the early 1970's, detectors that could continually monitor underground carbon monoxide became availible. They could be used to transmit carbon monoxide concentrations to the surface. The drawback of only being able to monitor CO led to the development of the tube bundle system.

The tube bundle system consists of small bore pvc tubes, through which mine air is continually drawn to the surface. A gas analyser on the surface analyses each sample tube in turn, recording the analyses (36). This provides a virtually continuous four gas analysis for many points in the mine. Flame, dust, and moisture traps are incorporated in the system. Significant changes in the mine atmosphere composition, and the Graham ratio can be spotted and pinpointed quickly.

The one great drawback to the tube bundle system is the time delay between sampling and analysis. The time between a sample entering the tube underground, and being analysed on the surface can be up to two hours. The system has become the most common method to detect heatings. A further advantage is that of remote monitoring. Tubes can be left in places inaccessible to personnel, sealed off districts for example, to sample the air. This aids the decision of when to enter an abandoned area.

A further problem is when tubes get broken. If this happens, mine air will continue to be drawn through the tube and analysed as if it were coming from the intended sample site, with no indication that there is a fault. For this reason the integrity of the tubes has to be regularly checked.

4.2.4 ELECTRONIC CONTINUOUS MONITORING

The steady progress of microelectronics, and the desire to have instant gas samples has led to the development of electrical transducers to measure gas concentrations (37). These transmit electrical signals to the surface for display, and analysis by computer systems. The transducers generally measure all oxidisable gases in a parameter called 'products of combustion'.

Eicker (38) describes the development of electronic carbon monoxide detection devices. These are coupled to computers on the surface. By suitably programming the computer, short term changes in carbon monoxide concentrations, caused by blasting and diesel engines, can be eliminated. This leaves the computer able to detect changes in the general carbon monoxide level, and abnormal rates of rise.

It is possible that this sort of system may form the basis for the MINOS system of the future. At present MINOS tends to be coupled with tube bundle systems. The electronic transducer based systems, using computers to process the data, will not provide the ideal instant response, but do hold the prospect of very fast mine gas analysis.

4.3 PREVENTION OF SPONTANEOUS HEATINGS

To prevent a spontaneous heating, conditions conducive to an outbreak must never be allowed to occur. The basic way of doing this is to stop the flow of oxygen rich air over broken coal. The method of prevention will depent on local experience. Precautions against heatings in roadways have been discussed in Chapter 3.3.

4.3.1 ROADSIDE PACKING METHODS

The role of roadside packs in preventing spontaneous combustion was discussed in Chapter 3.3.3. The roadside pack must support the waste side, and provide an impermeable barrier to prevent air entering the waste. The pack should be made of strong, inert materials.

4.3.2 ROADWAY AND STRATA SEALING

The two types of sealing are roadway sealing and strata sealing. The object of both roadway and strata sealing is to seal all places where air is likely to be feeding in, and places where products of combustion are appearing. There are many methods of sealing, and many different brands of sealants.

Sealants can be used to backfill shuttering in the roadway sides, and sprayed or hand plastered onto roadway surfaces. The types of sealant used are cementitious fillers, sand, clay and rubber. Cementitious fillers, common trade names being Hardstop,

Pozament, Mandoseal, are the most frequently used. Upgrading of these materials is a continuing process. Sealants can be pumped through packs and behind face supports to seal areas of the waste, depriving heatings of fresh air.

Grouting and bougeeing are the terms used for high pressure injection of sealants into the strata. Cracks and breaks in the strata have inert, sealant materials forced in to stop the passage of air. In bougeeing the sealant is a bentonite type mixture, which does not set and so flexes with the strata, or a setting material. Grouting has a cement or gypsum product forced into the strata, to fill cavities and cracks preventing airflow.

Nearly all materials shrink and crack on drying and with strata movements. Sealed areas need to be inspected and patched regularly to maintain effectiveness. In areas liable to spontaneous combustion it is usual to seal the roadway sides where a fault or other broken ground crosses the airway. It is common practice to only seal or bougee on the waste side of a roadway. Leakage paths exist from the ribside, and the floor may be cracked, allowing air into the waste, so the whole roadway needs to be sealed.

4.3.3 PRESSURE BALANCING

The flow of air underground is caused by pressure differences. The pressure difference is created by the mine fans and natural ventilating pressure, while the pressure distribution underground depends on the roadway resistances and the air quantity

flowing through them. The pressure difference around a district, as well as causing air to move around the roadways also causes migration through leakage paths in the waste. Sealing the waste prevents this airflow, pressure balancing reduces the pressure difference causing the leakage.

A pressure chamber consists of a fan and controlling air regulator. If sited in the return, a positive pressure is needed. This is produced by a forcing auxiliary fan. Pressure chambers in the intake require a negative pressure. This is produced with an exhausting fan.

Old workings are frequently at different pressures to present workings. When the two are in close proximity, seepage may occur from one to the other. Creating a pressure chamber behind a stopping alters the pressure and can stop the leakage. When a district has been abandoned and stopped off, there will be a pressure difference between the two stoppings. This will cause leakage around the old district. A simple pipe connection can be used to balance the pressure at the intake and return stoppings of an abandoned district.

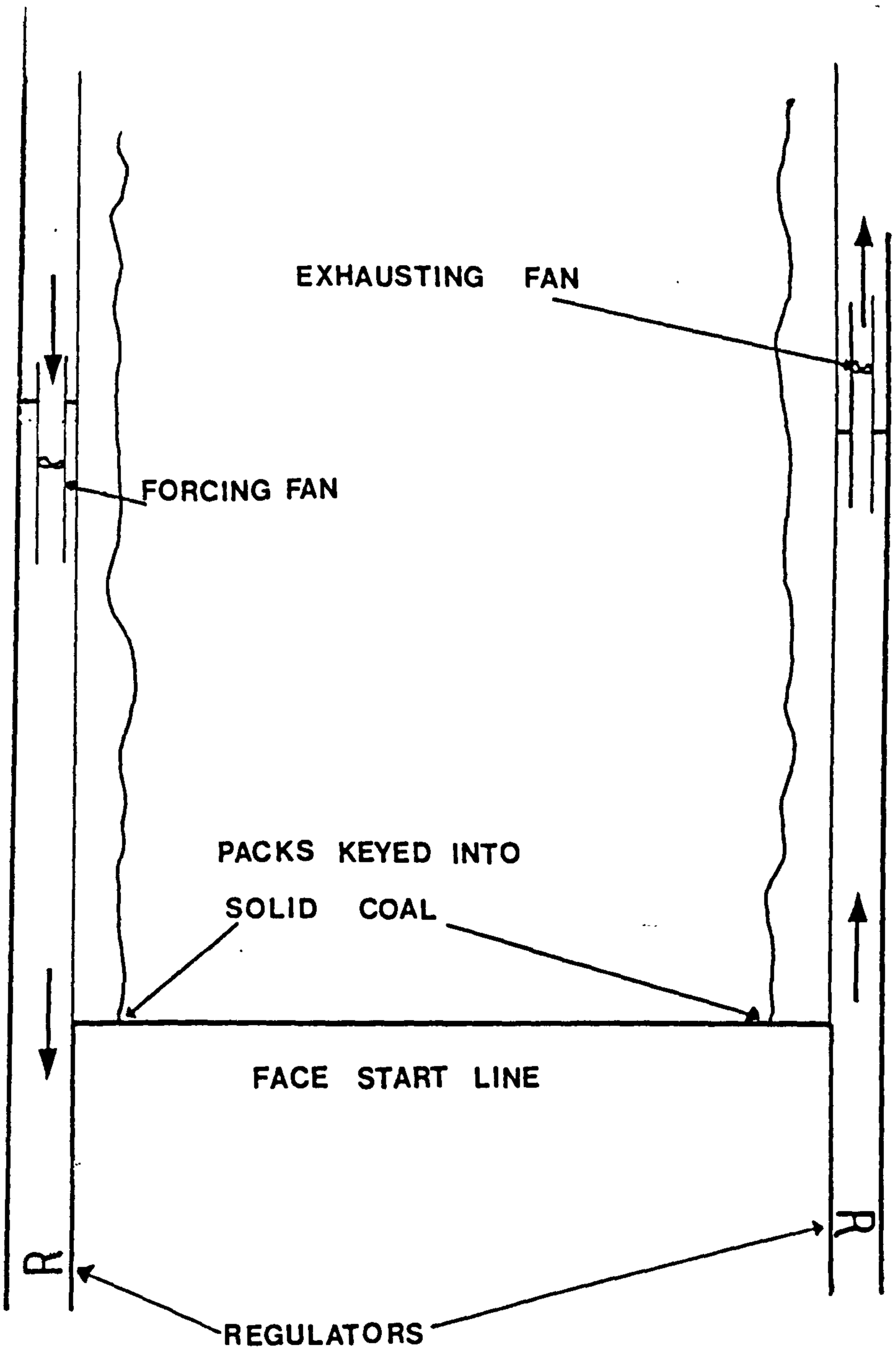
A common use for pressure chambers on working advance faces is the face start line. This has been done several times at Daw Mill (39). Once installed, the pressure chamber may have to be maintained for the life of the district. In Germany, pressure chambers have been used in conjunction with nitrogen injection on face start lines, to create a stable, inert atmosphere. (40) Typical pressure balancing arrangements are shown in Figure 4.1

On working districts pressure chambers can be a hinderance to the movement of men, supplies or coal. They are sometimes used as a temporary measure to buy time for roadway sealing work to be completed.

4.3.4 INERT GAS

Inert gas is more commonly used to combat heatings once detected, and the use of inert gas is discussed in the next section. A specific case of using inert gas as a preventative measure is detailed in Chapter 6. The objective of use is to flood the waste with inert gas, reducing the oxygen content sufficiently to inhibit spontaneous combustion. When inert gas is used as a precaution, it is used in smaller quantities than when combatting a heating. The gas is injected in the intake side at several sites, including likely leakage paths.

FIG. 4.1 PRESSURE BALANCING ON A FACE START LINE



4.4 COMBAT AGAINST SPONTANEOUS HEATINGS

Once it has been established that a heating has started, it is necessary to stop its progress to prevent an uncontrollable fire. Locating the position of a heating can be difficult especially with deep seated waste heatings. In most cases the main indication of a heating are the products of combustion. These can appear in several places, and may be some distance from the seat of the heating. It is important to establish the source of fresh air feeding the heating. In multi-lift workings this can prove especially difficult. Inspections of roadways for areas of broken ground or obstructions which constrict air flow should be undertaken. A ventilation pressure survey shows the pressure distribution round a district. From this it may be possible to locate places where the distribution gives rise to air leakage.

The method of dealing with a heating will depend on the location, size, and development of the heating, and local experience. Methods of combat are often the same as those used in prevention, such as roadway and strata sealing and pressure balancing. Such methods are mentioned above. The difference between prevention and combat usually lies in the intensity of the operation.

4.4.1 DIRECT ATTACK

This method can only be used when the heating is small and close to the roadway sides. The method is to dig into the roadway side to the heating and remove all the hot materials. The resulting void is filled with inert material, sand, hardstop or crushed rock are frequently used, while the dug out material is broken up and allowed to cool.

Another method of direct attack depends somewhat on luck. If the heating is intersected by a drill hole, water can be pumped in to extinguish it. The use of water may wash out dust and material filling cracks in such a way that air can travel in and cause another heating. The effect of moisture on the liability of a coal to heat has been examined earlier. This problem can be overcome by using sealants, after the heating is extinguished, to seal the strata.

The reaction of water on hot materials can produce water gas. This, being largely hydrogen, is highly explosive. Further dangers to workmen include carbon monoxide poisoning, heat exhaustion and fire if the hot material is at ignition temperature in fresh air.

Where the heating is deep seated, its location cannot be accurately determined or where it would be dangerous to dig out, the other combat methods have to be used.

4.4.2 SEALING

The best place to apply sealing techniques can be a matter of trial and error. The obvious places to inject the strata and seal the roadway sides are at leakage paths into the waste, and sites where combustion products are emitted into the roadway. Success has been achieved by sealing the intake roadway opposite the point in the return roadway where high carbon monoxide levels are detected in the pack sample pipes (32).

4.4.3 INERT GAS

The first reported use of inert gas to control a mine fire was in the middle of the last century (41). Inert gases are regularly used in the chemical industry to neutralise explosive gases. The use of very large carriers to transport petroleum around the world led to the development of inert gas generators. These can produce up to 16,000 cubic metres per hour of inert gas.

The mine atmosphere is made inert by introducing inert gas in order to reduce the oxygen content of the atmosphere around a fire, and to convert explosive mixtures of gas and air into non-explosive mixtures. A distinction has to be drawn between space and object inertisation.

Space inertisation means that an area which has been sealed off is filled with inert gas. This gas must be injected at the intake stopping. The method allows sealed off areas to be

recovered more quickly than when other methods are used.

Object inertisation means that the inert gas is applied to the seat of the fire. This method is used if a heating can be controlled without interrupting the district ventilation. On advance faces it is possible to inject gas into the goaf from the intake roadway. The migration of air towards the heating must be replaced by inert gas. This will be aided if additional sealing measures are taken. Up to the present time it has proved impossible to predict accurately the leakage paths, or the site of a heating. It is difficult to ensure that the inert gas displaces oxygen from reaching the heating, without using such large quantities that can prevent men from working in the return.

4.4.3.1 Choice Of Inert Gas

Carbon dioxide, nitrogen, and combustion gases have all been used to fight mine fires. In theory any inert gas could be used, but practical and cost limitations restrict the use.

Carbon dioxide is widely used in fire extinguishers, however for underground use it has several disadvantages:

i. It is not availible in large quantities.

ii. Under heating it can decompose to form carbon monoxide and oxygen.

iii. It is absorbed by the strata.

It has never been used in Britain, and only occasionally overseas.

Combustion gases are used in the oil industry to purge carrier ships. The gases are generated using an oil fired jet unit. Despite being used in Poland and other eastern bloc countries, (42) largely for space inertisation, this method has not found widespread approval elsewhere. There are reservations about the use of kerosene underground. The exhaust gases are hot, have a high relative humidity and contain high proportions of carbon dioxide, (up to 18%), carbon monoxide and other toxic gases. The method can only be used to fill an entire working district with inert gas.

Because of the disadvantages of other methods, nitrogen has become the most widely used inert gas in western Europe. Liquid nitrogen is delivered to the mine in tankers. An evaporator is used to produce nitrogen gas, which is carried underground in pipelines. Nitrogen has several advantages as an inert gas. It is easy to deliver on site, and is availible in large, but not unlimited quantities. A relatively pure, non-toxic, inert gas is produced, which because it is dry and cold improves the environmental conditions for firefighters. The major drawback to large scale use is the cost of gaseous nitrogen, plant hire and associated site preparation. The nitrogen supply cost is around 12p/cubic metre at 1985 prices (43). Efforts are being made in Germany to develop surface plant that separates nitrogen from air to reduce the cost (44).

Pipes are laid into the waste, from the intake roadway, as a face advances or retreats. When a heating develops, these pipes are coupled to the supply, and are used to flood the area with nitrogen. The first major use of nitrogen in the UK was in 1962 at Fernhill Colliery. Nitrogen was introduced behind the stoppings to enable recovery and sealing operations to be completed (45). It was in West Germany, though, that the large scale use of nitrogen was developed. From the first use in 1974, it has become part of normal mining operations to prepare for nitrogen injection in areas liable to spontaneous combustion (46). The first use of nitrogen in longwall waste in the UK was on 15's at Daw Mill in 1980 (41). Subsequently the method was adopted as a precautionary measure when salvaging districts in areas liable to spontaneous combustion (47).

In common with pressure chambers, nitrogen injection does not appear to extinguish a heating, but to suppress it. Evidence for the suppression of spontaneous combustion comes from the loss of 65's at Daw Mill. The face had to be sealed off, despite intense combat methods, including nitrogen injection. Once sealed the district was filled with nitrogen. Six weeks later there was no evidence of continuing fire behind the seals. The stoppings were broken down, and the district flushed with fresh air. On reentry the roadways were cool, and there were no signs of fire. Forty eight hours later the fire had restarted, and the face had to be sealed off again. There are other examples of cases where heatings have appeared to be under control, but when the nitrogen supply is reduced or discontinued the heating reappears. (30,39)

Very little is known about the flow of nitrogen in the waste, or the airflow and oxidation zones. Nitrogen is injected with little knowledge of where it is needed, or where it is going. Work on the movement of gases in longwall wastes should help determination of how nitrogen can be best used.

4.4.4 STOPPINGS

When all other methods fail to control a heating, the district affected has to be sealed off and abandoned. In cases where no recovery can be attempted, this will result in the loss of millions of pounds worth of equipment, and the sterilisation of reserves.

The requirements of stoppings are correct siting, good construction, rapid completion and simultaneous sealing.

The primary consideration when selecting a stopping site is the safety of the men building the stopping. Other factors are the ease with which materials can be transported to the site, and the facility with which the men can build the stopping. Primary stoppings are often situated at the end of a gate roadway. This does away with the need to ventilate a blind heading.

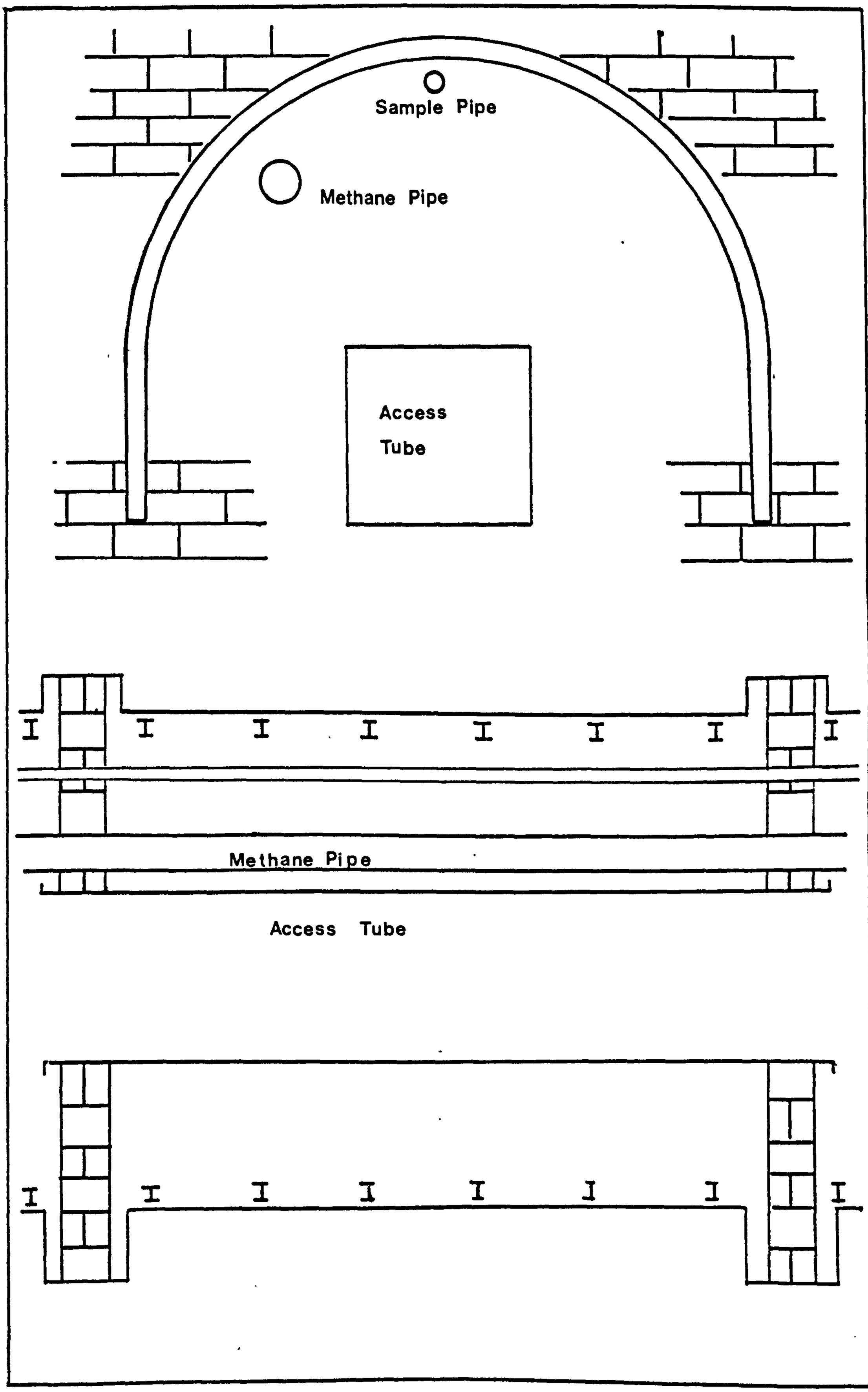
It is a sensible measure to prepare the stopping site. In seams not liable to spontaneous combustion this can consist of merely selecting the site, ensuring the roadway is in a good enough condition, and assembling a stock of materials. In seams that are liable, the stopping is sometimes partially built.

In order to provide an effective seal, the edges of the stopping must be keyed into the roadway sides, and the surrounding strata grouted. Stoppings should be in solid ground. Methods of construction vary. A common method used to be to build a stopping with dirt filled bags and plastered masonary walls. This provides an inexpensive stopping, but it is time consuming to build, difficult to seal simultaneously and does not give a perfect seal.

To combat this, prepared explosion proof stoppings have been developed. In the Western Area the stopping is partially completed, leaving a small access route sufficient for materials transport, men and salvage. Pozament is used to form the stopping, while the small hole left can be quickly bricked up and filled with cement materials. Stoppings in North Yorkshire incorporate a 2.2m square steel access tube, with explosion proof doors at each end, Figure 4.2. The doors frames and tube are surrounded by the agalite block wall and pump packing stopping. The doors can be quickly closed, after removing obstructions such as rails, and the tube filled. If the doors are not used they can be salvaged and taken to another district. It is obvious that this type of prepared stopping can be sealed almost simultaneously at intake and return, preventing an explosive atmosphere forming in the district while men are working to build the stoppings.

No stopping is totally airtight. Air will continue to flow in very small quantities around the district. Pressure balancing can help prevent this, and stoppings need to be regularly inspected to ensure they maintain effectiveness.

FIG.4.2 PARTIALLY BUILT STOPPING



4.5 CONCLUSIONS

The methods of detecting spontaneous combustion have been examined.

Many different measures can be taken to prevent and combat oxygen reaching areas liable, or subject to spontaneous combustion. The cost of prevention can be high, but the cost of not carrying out this work can be far greater if outbreaks have to be combatted, and workings abandoned.

Nitrogen injection to make inert the waste gas is a new technique in the UK. Nothing is known about the behaviour of the nitrogen once it is injected into the waste, or where heatings occur behind the face line. The next chapter examines an investigation that was conducted into the movement of nitrogen once injected into the waste of an advance face.

CHAPTER 5

NITROGEN FLOW MEASUREMENT THROUGH THE GOAF

5.1 INTRODUCTION

The use of nitrogen to produce an inert atmosphere in an area of active or potentially active spontaneous combustion is not new. It has been reintroduced to this country in the last few years. The first application of the technique to a longwall coalface waste in the UK was in the South Midlands Area, on 15's face at Daw Mill Colliery (41). Geological difficulties had slowed production virtually to a halt, and during this period the carbon monoxide level started to rise uncontrollably. The use of nitrogen enabled the situation to be controlled. Since this time nitrogen has been used on several faces, either as a precautionary measure, or because a heating was active.

The loss of 65's face at Daw Mill made it apparent that little was known about the gas flow through a coal face goaf (30). Nitrogen was first used on 65's when production had been interrupted and carbon monoxide levels were rising. The heating was controlled,

and the carbon monoxide level brought down to, and maintained at, an acceptable level. The ventilating pressure of the district was reduced, and nitrogen continued to be injected into the goaf. After four months the carbon monoxide levels started to rise again, and it proved impossible to control the heating, despite increasing the nitrogen flow, attempting to inject the nitrogen near to where it was thought the heating lay, and reducing the airflow around the district. The oxygen level in the return fell so low that men could not enter this part of the district to seal the roadway sides. Eventually the district had to be sealed off.

During this time the difficulty in determining the site of the heating, and the behaviour of the nitrogen once injected into the waste were felt to be key factors in the loss of the face. The experiment at Daw Mill was arranged to investigate the flow of nitrogen through the waste.

This chapter details an investigation that was carried out into the behaviour of nitrogen that was injected into the waste of an advance face to reduce the level of oxygen in the waste gas, and hence reduce the liability to spontaneous combustion.

5.2 OBJECTIVES

1. To obtain information about the path of nitrogen through the waste.

2. Improve the utilisation of nitrogen.

3. Gain knowledge about the passage of other mine gases in caved wastes.

5.3 THE SITE

Daw Mill Colliery is in the Warwickshire coalfield, midway between Birmingham and Coventry. Coal is mined from the Warwickshire Thick Coal seam, which is extremely liable to spontaneous combustion. There have been many incidents of spontaneous combustion heatings since the colliery was opened in 1965, of which at least seven have resulted in the loss of a face. (39) Despite precautions, successful salvage of face equipment was a rare event. With the arrival of chock shield supports it was decided that salvage would be desirable, and could be possible using nitrogen injection. A stationary face, with nitrogen being introduced as a precautionary measure gave satisfactory conditions for a trial to examine the movement of injected nitrogen.

The face chosen for the experiment was 15's in the Warwickshire Thick Coal at Daw Mill. The district was being salvaged, having advanced 1279m. This was a 230m long advance face, designed to extract 3.5m of coal, fitted with heavy duty production equipment. There had been two previous incidents of spontaneous combustion during production, both of which had been successfully controlled. As mentioned earlier, it was the first face in the UK on which nitrogen was injected into the waste to combat a heating.

A nitrogen range remained in place from one of these outbreaks. The expected duration of salvage exceeded the normal incubation time for the coal. The usual precautions prior to salvage were implemented, fully sealing the roadway side for the last 100m of the face run, sealing the intake pack area and ensuring that all packs were completed to a high standard. In addition nitrogen injection was used. This was started in August 1983, 8 weeks before the face reached the pre-planned stop line.

Nitrogen was being introduced for 12 weeks prior to the commencement of the experiment. According to the theories current, the nitrogen flow through the waste should have stabilised into a constant pattern, while the nitrogen content of the general body ventilation should gradually increase along the face to the return end. (30)

Nitrogen was injected into the waste as a precautionary measure from the intake roadway through five holes. Each hole was bored through the pump pack which formed the gateside pack, and lined with a stand pipe. Nitrogen was pumped from a surface evaporation plant via a steel range to a manifold, just outbye of the furthest hole from the face, for distribution and flow measurement. Flow measurement was obtained by an orifice plate and pressure tappings at the manifold for each hole. The nitrogen passed through ordinary firehoses with standard connections to the pipe.

The holes used to inject nitrogen were at the following positions, and the flow of nitrogen to each hole is shown.

Metre Mark	Quantity of Nitrogen m^3/min
1040	2.3
1120	1.9
1170	1.0
1210	1.1
1255	2.3
Total	8.6

The ventilation around the district during the experiment was $6m^3/s$ of air.

5.4 PROCEDURE

The injection of nitrogen into the waste disrupts the normal goaf flow patterns. It would clearly be impossible to monitor the flow from every hole, while if the flow from one hole only was monitored, this flow would likely be distorted by the effects of flow from the other injection holes. The sampling of the goaf gases would be a short distance into the waste, one or two metres, so no information about the gas flow patterns deep in the waste could be directly obtained. It was decided to monitor the flow from the furthest outbye hole, at the 1040 metre mark. In order to identify this flow and separate it from the ventilating air and other nitrogen, a tracer technique was used.

Sulphur hexafluoride has been used in this country and many others as a tracer for underground ventilation work as it is colourless, odourless and non-toxic. At normal temperatures it is a vapour, it can be compressed into a liquid. The concentration of sulphur hexafluoride can be determined using a gas chromatograph with an electron capture head. Concentrations as low as 1 part in 10^{12} can be detected. The British Coal regional laboratories at Mansfield Woodhouse are equipped to carry out this analysis.

In order to ensure that the tracer gas did not escape from the injection system, the tracer was introduced as close as possible to the hole. A special adaptor was made at the colliery so that the gas could be injected into the hole, a photograph is shown in Figure 5.1

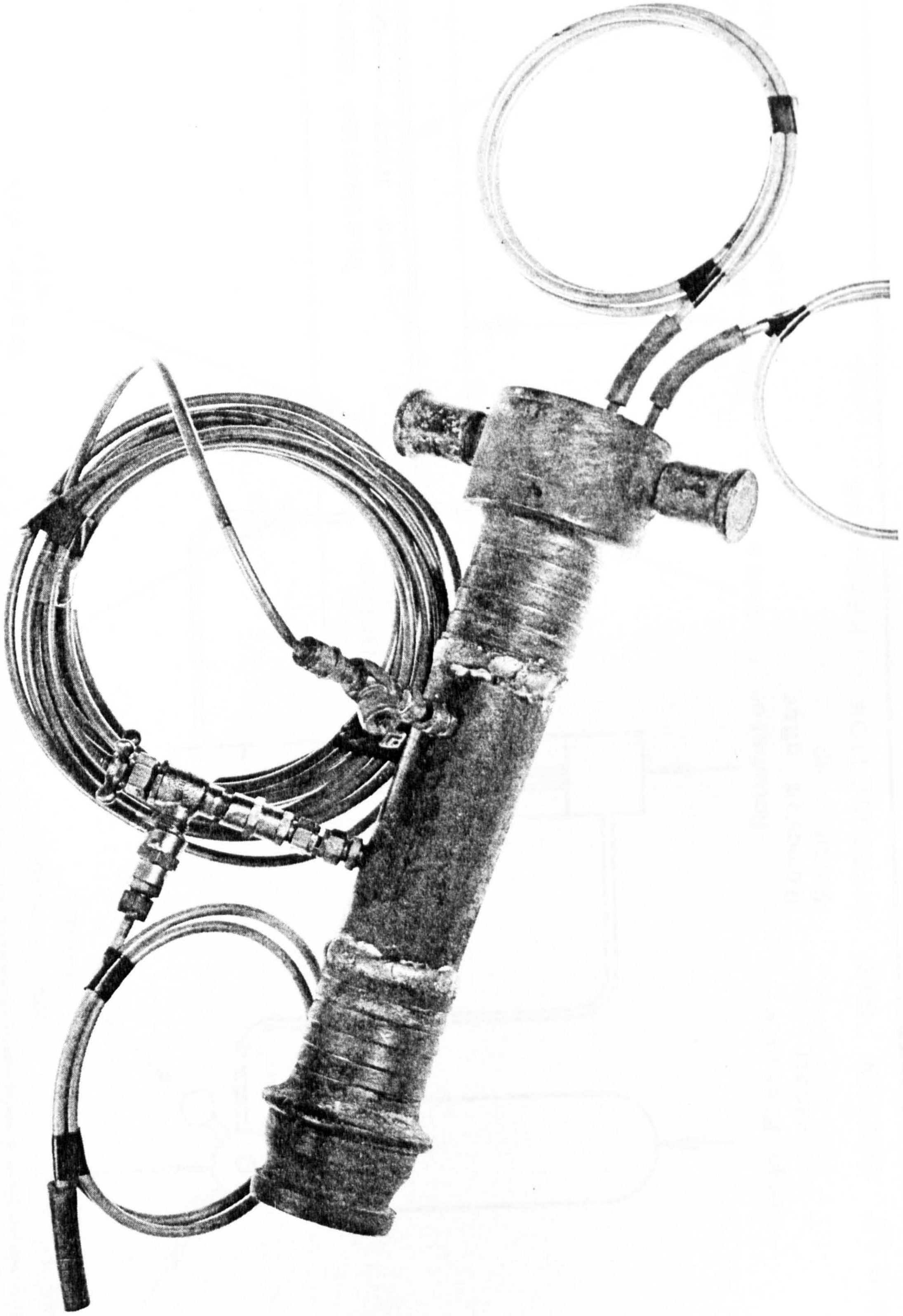
A mixture of approximately 1% sulphur hexafluoride and 99% nitrogen was prepared and contained at 143 bar in a 7 litre high pressure cylinder, which yields 1,000,000 ml at atmospheric pressure. The cylinder was connected by a pressure reducer to a needle valve, through to a rotameter, and then onto the adaptor piece which had been inserted into the nitrogen line. The regulator was set to obtain a flow of 120ml/min of the tracer gas, giving a flow sufficient for over five days. So that this could not be tampered with underground, the normal handle on the regulator was removed and replaced by a screw. Once the required flow had been obtained the rotameter was removed from the circuit, and the tracer allowed to to flow into the nitrogen being introduced into the waste. Figure 5.2 shows the arrangement of the tracer gas

introduction apparatus when the flow was being set up, and Figure 5.3 the position of the various components in the roadway which became important during the time measurements were being taken.

5.5 SAMPLING PROCEDURE

The sample points were established in the week prior to the installation of the sulphur hexafluoride injection equipment. Eleven holes were bored between the face supports, 1.5m into the waste, using casing rods with a drill bit welded onto the end of the rod. Another casing rod was screwed onto the drilled rod in order to ensure that the casing would conveniently reach the travelling way in the chock shield supports. Into this was inserted a length of 'gripflex' pvc tube with an internal diameter of 5mm. The end of the casing rod was sealed with hard stop. This apparatus is shown is Figure 5.4

FIGURE 5.1 SF_6 INTRODUCTION AND SAMPLING EQUIPMENT



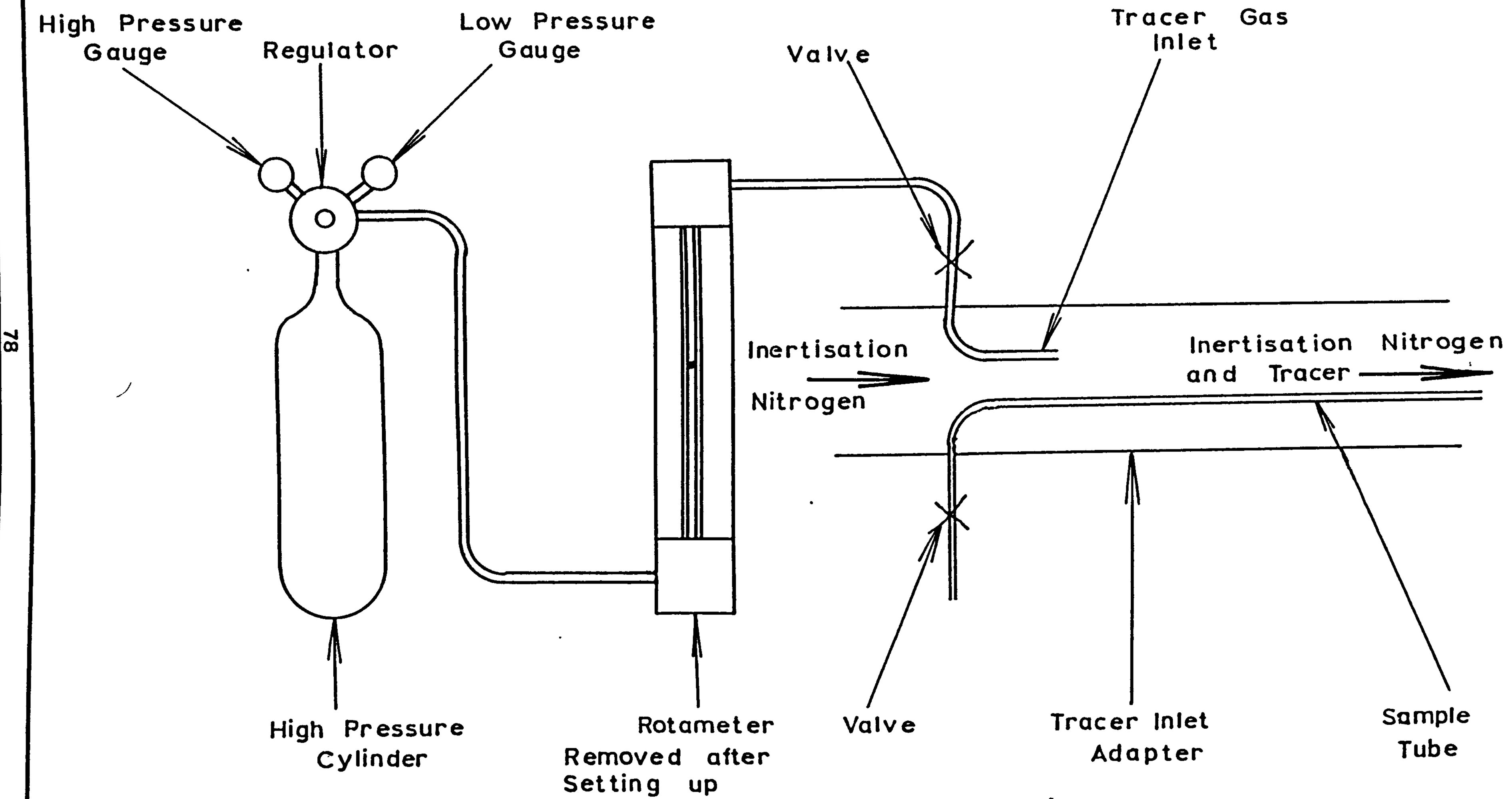


FIG. 5.2 TRACER GAS INTRODUCTION APPARATUS

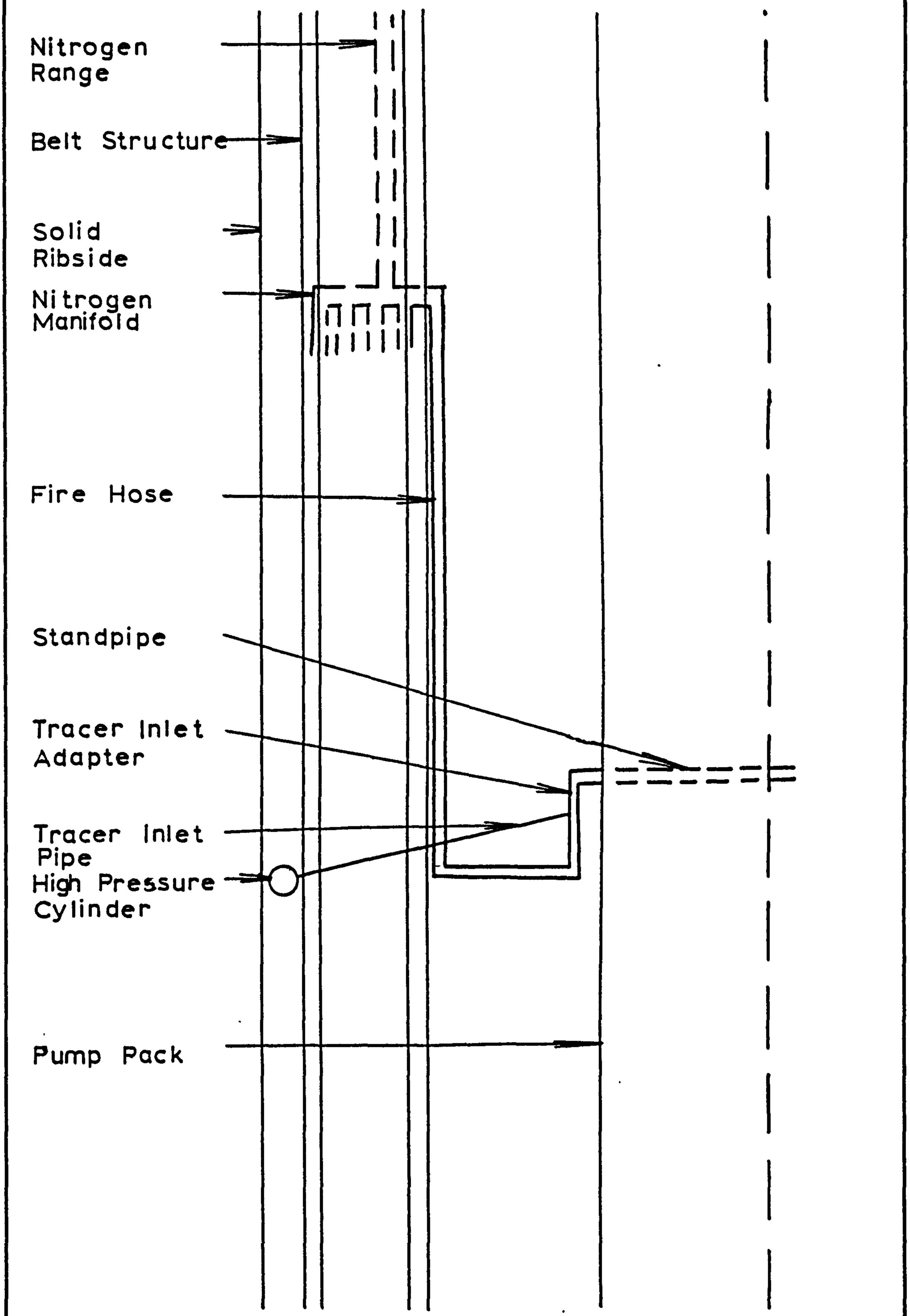


FIG. 5.3 POSITION OF APPARATUS IN ROADWAY

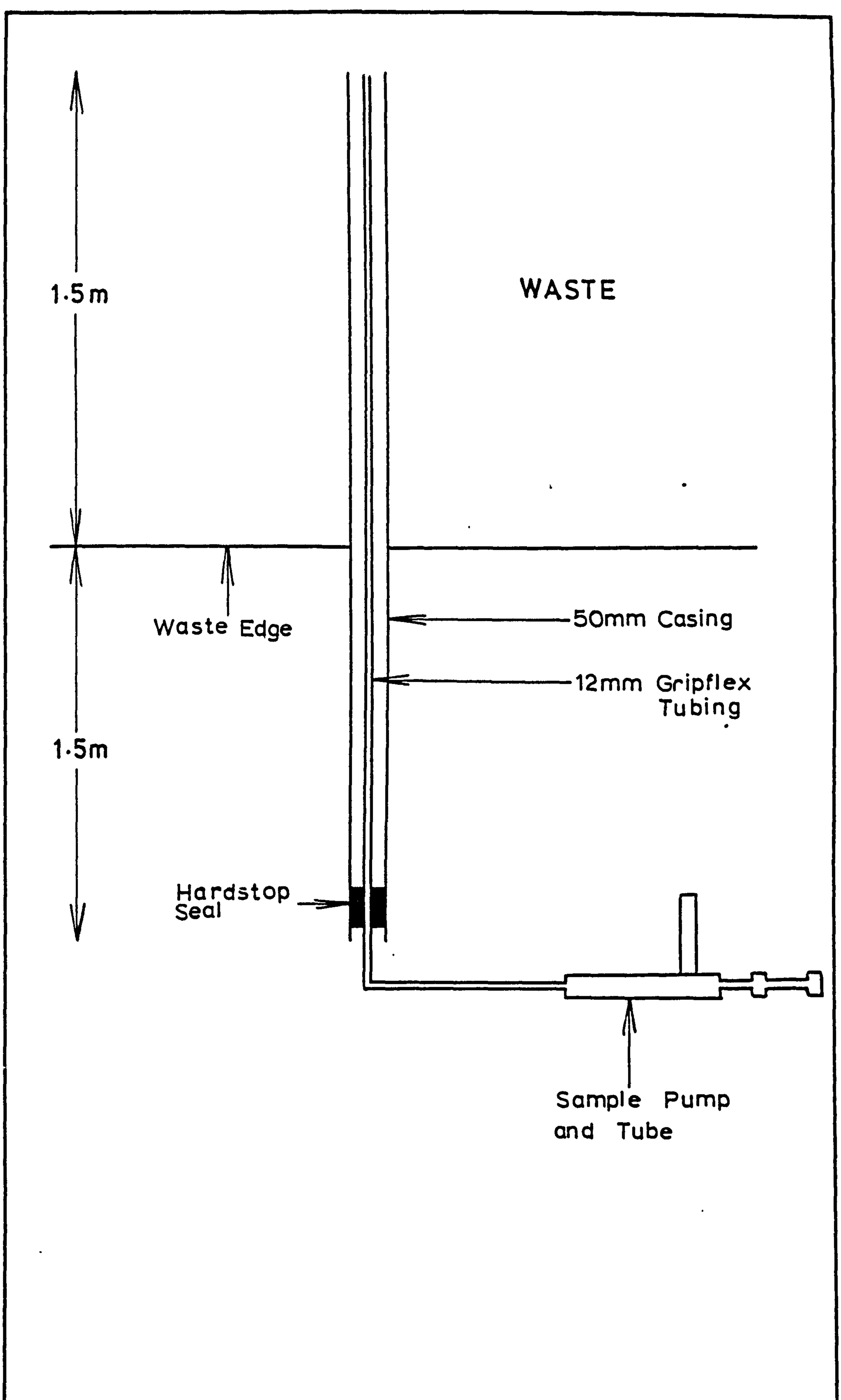


FIG.5.4 SAMPLING POINT

The positions of the pipes were at the following chocks;

5 - 6 (packside)	84 - 85
18 - 19	94 - 95
34 - 35	108 - 109
47 - 48	123 - 124
56 - 57	140 - 141 (packside)
69 - 70	

In addition to this the following general body sample positions were used:

1. 940m in the intake, 100m outbye of the injection point
2. 10m outbye of the intake rip
3. 10m outbye of the return rip
4. Outbye end of the return, using the tube bundle system

Figure 5.5 shows the position of each of the sample points. Regular samples had to be taken. It was felt that four hourly samples were needed, that is twice a shift, at each of the face sample points, while the district general body samples were taken at the start and finish of each shift. Three Mining Engineering Trainees were used to take the samples, after being instructed in the sampling techniques to be used.

In order to take accurate samples from the waste pipes hand held SMRE sample pumps were used.

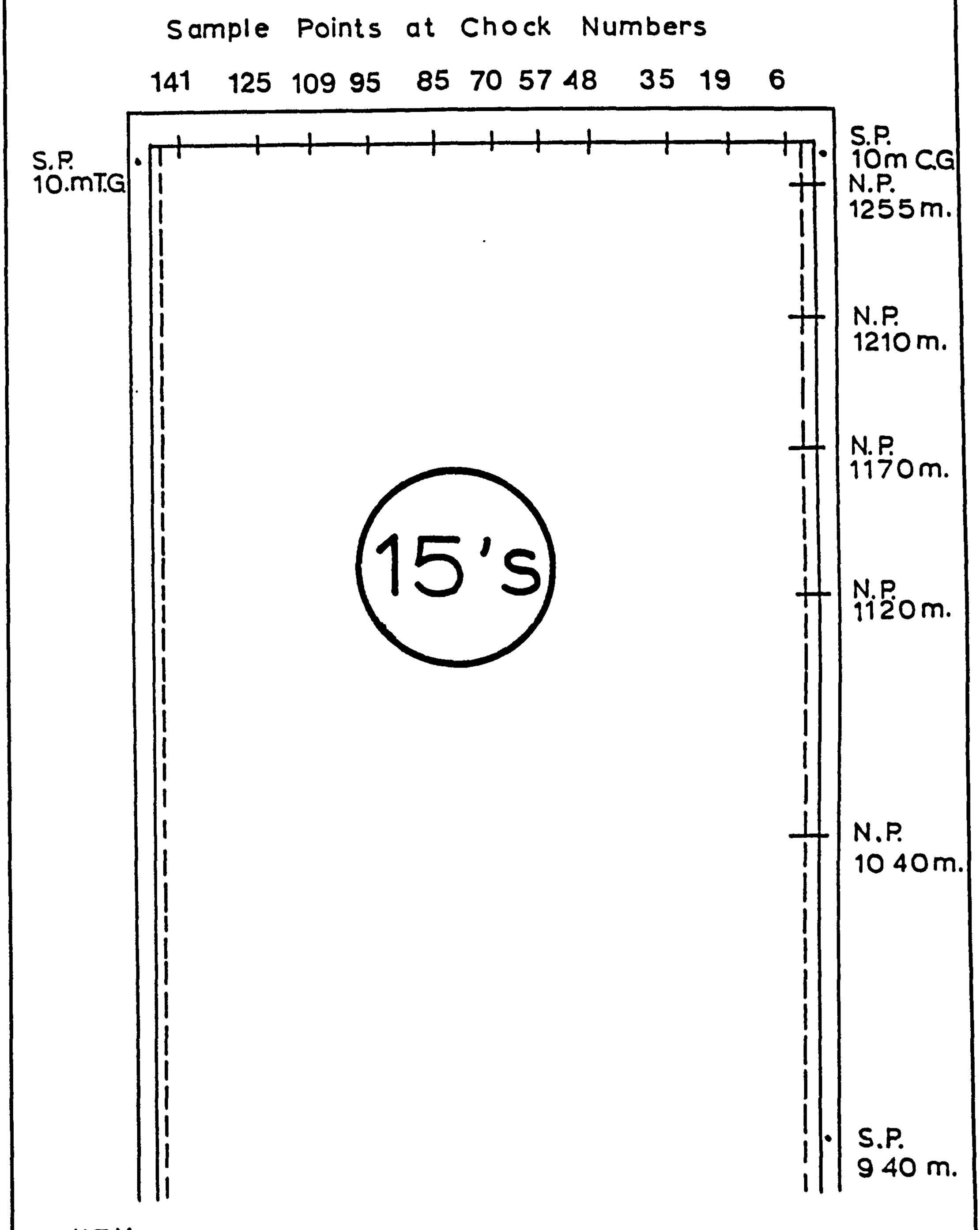


FIGURE 5.5 15's DISTRICT SAMPLING POINTS

The tube was first purged by connecting the sample pump and pumping several times. The sample collection tube was attached to the pump, filled and emptied to purge it, and then the sample was taken. The tubes were taken daily to the laboratories at Mansfield Woodhouse for analysis.

5.6 RESULTS

In the space of a week some 500 gas sample were collected and analysed. The large number of samples leads to problems in the presentation and interpretation of data. It was found that the tube bundle sampling system was not working properly, so there are no results for the outbye end of the return.

The raw data is in Tables 1 to 14. Plots of the oxygen, SF_6, carbon monoxide and barometric pressure against time at each sample point are shown in Figure 6 to 19.

It very quickly became apparent that the SF_6 levels in the intake were far greater than those which had been expected. A series of samples extra to the normal samples was taken immediately around the nitrogen/SF_6 injection point and SF_6 cylinder, and along the intake roadway inbye of the injection point. The gas analysis of these samples in in Table 15.

5.6.1 FACE PROFILES

To reveal the level of flow through the goaf to the face, gas analysis was plotted against distance along the face, shown in Figures 20 to 46. There are two sets of graphs. One shows oxygen and SF_6 levels, and the other carbon monoxide levels. The time given on each graph is the median sampling time, as it is not possible to simultaneously sample each point along the face. The spread of sample times was about 20 minutes each side of the median. This is satisfactory, except when the barometric pressure was changing rapidly, such as the afternoon and evening of January 27th.

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 940m INTAKE

DAY,TIME	CO2%	CH4%	O2%	CO%	SF6*10^12
23, 11.30	0.05	0.03	20.80	0.0002	100
23, 16.15	0.04	0.03	20.81	0.0110	72
24, 0.14	0.05	0.02	20.80	0.0003	0
24, 8.34	0.06	0.02	20.83	0.0003	50
24, 16.50	INSUFFICIENT SAMPLE TAKEN				40
25, 1.34	0.05	0.01	20.80	0.0005	0
25, 10.30	0.04	0.02	20.77	0.0004	50
25, 12.30	0.05	0.02	20.83	0.0001	40
25, 17.50	0.04	0.02	20.80	0.0002	30
26, 0.30	0.06	0.02	20.79	0.0003	0
26, 10.15	0.03	0.01	20.80	0.0002	55
26, 16.59	0.06	0.02	20.79	0.0003	0
27, 1.00	0.05	0.01	20.79	0.0005	0
27, 8.30	0.06	0.01	20.78	0.0003	0
27, 16.39	0.04	0.01	20.80	0.0001	35
28, 0.49	0.06	0.01	20.79	0.0001	30

Table 5.1 Gas Samples 940m Intake

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT,10 YDS OB COAL GATE RIP

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 12.34	0.07	0.06	20.28	0.0004	79	2420
23, 16.34	0.03	0.01	20.31	0.0002	79	1800
24, 0.25	0.05	0.03	20.34	0.0003	79	1320
24, 8.54	0.06	0.02	20.48	0.0003	79	1062
24, 16.30	0.06	0.01	20.44	0.0002	79	975
25, 1.15	0.08	0.02	20.41	0.0004	79	850
25, 9.29	0.05	0.02	20.36	0.0002	79	1332
25, 17.36	0.08	0.02	20.34	0.0002	79	1140
26, 1.00	0.05	0.03	20.37	0.0003	79	1584
26, 9.05	0.05	0.03	20.18	0.0002	79	1716
26, 18.00	0.07	0.02	20.43	0.0002	79	1800
27, 0.49	0.06	0.02	20.40	0.0002	79	891
27, 8.45	0.26	0.08	20.39	0.0002	79	561
27, 16.33	INSUFFICIENT SAMPLE TAKEN					972
28, 0.59	0.02	0.02	20.38	0.0005	79	820

NITROGEN BY DIFFERENCE

Table 5.2 Gas Samples 10m Sample Point, Intake Rip

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 6 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 12.39	0.05	0.05	10.39	0.0013	89	300
23, 17.10	0.05	0.04	10.89	0.0013	89	1404
23, 18.25	0.06	0.05	11.26	0.0013	88	1422
24, 2.30	0.10	0.04	12.66	0.0012	87	770
24, 4.50	0.06	0.02	12.58	0.0013	87	880
24, 9.00	0.06	0.02	13.24	0.0012	86	720
24, 10.59	0.05	0.02	13.44	0.0010	86	720
24, 16.59	INSUFFICIENT SAMPLE TAKEN					624
24, 19.39	0.05	0.02	12.89	0.0010	87	640
25, 1.00	INSUFFICIENT SAMPLE TAKEN					570
25, 3.55	0.10	0.04	13.06	0.0014	86	510
25, 9.20	0.05	0.02	12.32	0.0012	87	720
25, 11.39	0.05	0.02	13.69	0.0009	86	650
25, 17.34	0.05	0.02	12.05	0.0011	87	540
25, 20.30	0.05	0.03	12.66	0.0011	87	540
26, 0.55	0.05	0.03	12.50	0.0014	87	808
26, 4.04	0.07	0.07	12.58	0.0013	87	870
26, 9.00	0.04	0.03	12.66	0.0016	87	930
26, 11.30	INSUFFICIENT SAMPLE TAKEN					1404
26, 16.50	INSUFFICIENT SAMPLE TAKEN					680
26, 20.27	0.06	0.04	12.63	0.0016	87	578
27, 0.45	0.05	0.03	12.66	0.0019	87	660
27, 3.45	0.11	0.03	13.87	0.0016	85	611
27, 8.39	0.06	0.02	14.98	0.0017	84	693
27, 10.59	0.07	0.02	15.25	0.0020	84	627
27, 16.30	0.05	0.02	13.73	0.0013	86	612
28, 1.09	0.06	0.02	15.31	0.0018	84	648

NITROGEN BY DIFFERENCE

Table 5.3 Gas Samples at 6 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 19 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 12.49	0.09	0.07	20.06	0.0010	79	2200
23, 17.15	INSUFFICIENT SAMPLE TAKEN					1548
23, 18.25	0.04	0.05	20.08	0.0012	79	1638
24, 0.34	0.07	0.03	20.02	0.0012	79	1386
24, 3.40	0.12	0.04	20.10	0.0012	79	1320
24, 9.09	0.11	0.03	20.17	0.0009	79	1080
24, 11.04	0.07	0.02	20.22	0.0009	79	1152
24, 17.10	0.06	0.02	20.29	0.0009	79	819
24, 19.49	0.06	0.02	20.21	0.0008	79	1053
25, 0.49	INSUFFICIENT SAMPLE TAKEN					950
25, 3.49	0.07	0.02	20.73	0.0011	79	875
25, 9.14	0.06	0.02	20.24	0.0009	79	864
25, 11.45	0.06	0.03	20.23	0.0007	79	896
25, 17.30	0.05	0.02	20.26	0.0007	79	930
25, 20.27	0.06	0.02	20.30	0.0007	79	990
26, 0.29	INSUFFICIENT SAMPLE TAKEN					1485
26, 3.59	INSUFFICIENT SAMPLE TAKEN					1450
26, 8.54	0.04	0.02	20.11	0.0010	79	1443
26, 11.34	0.04	0.02	20.01	0.0010	79	1482
26, 16.45	0.07	0.04	20.19	0.0009	79	1156
26, 20.25	0.08	0.03	20.23	0.0009	79	816
27, 0.42	0.07	0.03	20.17	0.0008	79	1023
27, 3.40	0.06	0.03	20.23	0.0010	79	792
27, 8.34	0.07	0.02	20.23	0.0008	79	825
27, 11.04	0.06	0.02	20.33	0.0011	79	726
27, 16.24	0.06	0.03	20.23	0.0007	79	828
28, 1.24	0.06	0.02	20.26	0.0010	79	720

NITROGEN BY DIFFERENCE

Table 5.4 Gas Samples at 19 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 35 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 12.59	0.20	0.11	17.53	0.0075	82	1500
23, 17.13	0.16	0.10	18.43	0.0063	81	1512
23, 18.34	0.18	0.10	18.93	0.0055	80	1800
24, 1.59	0.16	0.06	19.22	0.0047	80	1260
24, 4.44	0.15	0.06	19.46	0.0041	80	1100
24, 9.14	0.22	0.05	19.72	0.0035	80	1134
24, 11.10	0.15	0.04	19.73	0.0033	80	1062
24, 17.25	INSUFFICIENT SAMPLE TAKEN					1001
24, 19.50	0.11	0.03	19.73	0.0031	80	1053
25, 0.43	0.12	0.04	19.77	0.0034	80	884
25, 3.39	0.13	0.04	19.60	0.0039	80	891
25, 9.09	0.12	0.02	19.77	0.0033	80	1008
25, 11.50	0.12	0.03	19.65	0.0036	80	1044
25, 17.25	0.13	0.04	19.53	0.0037	80	930
25, 20.25	0.14	0.04	19.50	0.0038	80	960
26, 0.45	0.15	0.06	19.15	0.0043	80	1287
26, 3.55	0.16	0.08	18.66	0.0045	81	1452
26, 8.50	0.12	0.06	18.74	0.0048	81	1482
26, 11.45	0.13	0.06	18.76	0.0045	81	1521
26, 16.42	0.32	0.10	18.84	0.0040	80	1275
26, 20.21	0.18	0.07	18.93	0.0044	80	1292
27, 0.37	0.16	0.07	19.12	0.0045	80	1056
27, 3.37	0.50	0.14	18.97	0.0043	80	990
27, 8.30	0.34	0.09	19.23	0.0041	80	891
27, 11.10	0.16	0.05	19.28	0.0039	80	990
27, 16.20	0.17	0.05	19.35	0.0038	80	1008
28, 1.30	0.17	0.04	19.56	0.0037	80	976

NITROGEN BY DIFFERENCE

Table 5.5 Gas Samples at 35 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 48 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 12.59	0.26	0.28	14.95	0.0075	84	1000
23, 17.19	0.26	0.29	16.19	0.0073	83	1548
23, 18.34	0.22	0.21	16.96	0.0096	82	1440
24, 0.45	0.25	0.18	17.55	0.0068	82	1278
24, 3.45	0.22	0.13	18.34	0.0063	81	1224
24, 9.20	0.20	0.07	19.38	0.0048	80	990
24, 11.15	0.20	0.07	19.48	0.0045	80	1026
24, 17.30	INSUFFICIENT SAMPLE TAKEN					1000
24, 19.54	INSUFFICIENT SAMPLE TAKEN					1053
25, 0.47	0.05	0.05	19.53	0.0044	80	891
25, 3.25	0.14	0.05	19.58	0.0044	80	850
25, 9.00	0.14	0.05	19.68	0.0039	80	1152
25, 12.00	0.14	0.04	19.77	0.0038	80	936
25, 17.21	0.13	0.04	19.56	0.0038	80	1110
25, 20.19	0.16	0.05	19.52	0.0041	80	900
26, 0.40	0.14	0.07	19.18	0.0047	80	1188
26, 3.49	INSUFFICIENT SAMPLE TAKEN					1518
26, 8.45	0.13	0.11	18.50	0.0057	81	1482
26, 11.50	INSUFFICIENT SAMPLE TAKEN					1443
26, 16.39	0.20	0.10	18.67	0.0057	81	1258
26, 20.19	0.18	0.09	18.87	0.0057	80	1258
27, 0.34	0.17	0.08	19.09	0.0053	80	1089
27, 3.31	0.35	0.14	19.10	0.0051	80	1023
27, 8.25	0.16	0.06	19.51	0.0045	80	924
27, 11.15	INSUFFICIENT SAMPLE TAKEN					800
27, 16.15	0.14	0.05	19.60	0.0040	80	900
28, 1.34	0.14	0.04	19.62	0.0044	80	829

NITROGEN BY DIFFERENCE

Table 5.6 Gas Samples at 48 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 57 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.04	0.11	0.11	19.33	0.0077	80	1800
23, 17.19	0.06	0.08	19.65	0.0054	80	1548
23, 18.45	0.06	0.07	19.98	0.0026	79	1440
24, 1.55	0.08	0.05	19.74	0.0048	80	1225
24, 4.39	0.08	0.06	19.82	0.0040	80	1206
24, 9.25	0.10	0.04	20.06	0.0030	79	1044
24, 11.19	0.08	0.03	20.11	0.0026	79	1170
24, 17.33	0.07	0.02	20.11	0.0023	79	975
24, 19.59	0.06	0.02	20.10	0.0019	79	1014
25, 0.32	0.09	0.05	20.04	0.0032	79	680
25, 3.29	0.10	0.03	19.96	0.0037	79	782
25, 8.54	0.06	0.02	19.98	0.0023	79	1116
25, 12.09	0.06	0.02	20.04	0.0023	79	1008
25, 17.17	0.03	0.02	19.71	0.0023	80	1200
25, 20.17	0.08	0.03	20.10	0.0025	79	960
26, 0.00	0.08	0.05	19.91	0.0037	79	1386
26, 3.42	0.08	0.06	19.80	0.0037	80	1584
26, 8.39	0.04	0.04	19.76	0.0035	80	1404
26, 11.54	0.04	0.04	19.75	0.0037	80	1258
26, 16.39	0.20	0.10	18.67	0.0057	81	1404
26, 20.19	0.18	0.09	18.87	0.0057	80	1258
27, 0.31	0.23	0.08	19.85	0.0036	79	1122
27, 3.28	0.30	0.12	19.93	0.0032	79	891
27, 8.19	0.11	0.03	19.98	0.0029	79	1056
27, 11.19	0.08	0.03	20.09	0.0026	79	792
27, 16.07	0.08	0.03	19.93	0.0033	79	828
28, 1.39	0.10	0.04	19.67	0.0059	80	792

NITROGEN BY DIFFERENCE

Table 5.7 Gas Samples at 57 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 70 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.04	0.12	0.11	19.87	0.0022	79	1760
23, 17.25	0.05	0.10	20.01	0.0014	79	1872
23, 18.40	0.07	0.10	19.71	0.0031	80	1458
24, 0.55	0.07	0.07	20.04	0.0012	79	1350
24, 3.55	0.07	0.06	20.13	0.0011	79	1170
24, 9.29	0.09	0.04	20.25	0.0011	79	990
24, 11.25	0.10	0.04	20.21	0.0012	79	1116
24, 17.34	0.08	0.02	20.17	0.0011	79	975
24, 20.02	0.07	0.03	20.10	0.0012	79	1014
25, 0.29	0.08	0.04	20.12	0.0013	79	825
25, 3.25	INSUFFICIENT SAMPLE TAKEN					1000
25, 8.50	0.06	0.02	20.21	0.0010	79	1332
25, 12.14	0.07	0.03	20.11	0.0011	79	1008
25, 17.15	0.07	0.05	20.10	0.0010	79	1200
25, 20.14	0.08	0.05	20.06	0.0012	79	960
25, 20.14	INSUFFICIENT SAMPLE TAKEN					1370
26, 3.37	0.03	0.09	19.90	0.0015	79	1617
26, 8.34	INSUFFICIENT SAMPLE TAKEN					1443
26, 12.00	0.07	0.08	19.87	0.0015	79	1482
26, 16.34	0.09	0.05	19.97	0.0026	79	1258
26, 20.15	0.08	0.08	20.08	0.0017	79	850
27, 0.29	0.09	0.07	20.01	0.0014	79	1089
27, 3.25	0.08	0.05	20.10	0.0016	79	891
27, 8.14	0.08	0.05	19.98	0.0015	79	990
27, 11.30	0.08	0.04	20.14	0.0014	79	759
27, 16.09	INSUFFICIENT SAMPLE TAKEN					1188
28, 1.44	0.07	0.03	20.10	0.0015	79	792

NITROGEN BY DIFFERENCE

Table 5.8 Gas Samples at 70 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 85 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.09	0.57	0.53	16.89	0.0120	81	2000
23, 17.25	0.25	0.28	18.10	0.0130	81	1512
23, 18.45	0.21	0.25	18.78	0.0069	80	1800
24, 1.04	0.22	0.21	18.90	0.0066	80	1350
24, 4.00	0.22	0.21	18.81	0.0082	80	1100
24, 9.40	0.25	0.14	19.24	0.0064	80	1026
24, 11.30	0.24	0.14	16.39	0.0061	83	1062
24, 17.39	0.19	0.14	19.21	0.0064	80	1035
24, 20.07	0.18	0.16	19.20	0.0056	80	1014
25, 0.25	0.22	0.20	18.98	0.0072	80	850
25, 3.20	0.21	0.14	19.05	0.0070	80	858
25, 8.45	0.21	0.11	19.01	0.0074	80	900
25, 12.20	0.21	0.11	19.08	0.0067	80	1088
25, 17.13	0.19	0.13	18.97	0.0059	80	1260
25, 20.13	0.19	0.20	18.87	0.0061	80	960
26, 0.25	0.18	0.20	18.27	0.0079	81	1172
26, 3.29	0.21	0.25	18.04	0.0079	81	1403
26, 8.30	0.19	0.23	18.07	0.0110	81	1404
26, 12.05	0.13	0.26	18.34	0.0080	81	1404
26, 16.32	0.10	0.08	19.96	0.0014	79	1105
26, 20.13	INSUFFICIENT SAMPLE TAKEN					1226
27, 0.25	0.20	0.24	18.69	0.0080	80	957
27, 3.20	0.22	0.20	18.83	0.0076	80	9957
27, 8.10	0.21	0.18	18.98	0.0071	80	858
27, 11.34	0.22	0.18	19.07	0.0063	80	891
27, 16.07	0.21	0.20	18.80	0.0073	80	936
28, 1.50	0.22	0.13	18.63	0.0010	81	936

NITROGEN BY DIFFERENCE

Table 5.9 Gas Samples at 85 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT, 95 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.09	0.11	0.16	19.62	0.0025	80	1980
23, 17.34	0.11	0.18	19.57	0.0026	80	1710
23, 18.45	0.14	0.20	19.14	0.0046	80	1530
24, 1.09	0.09	0.07	19.83	0.0021	80	1530
24, 4.04	0.08	0.07	19.97	0.0020	79	1314
24, 9.45	0.08	0.05	20.16	0.0019	79	1098
24, 11.34	0.08	0.04	20.18	0.0015	79	1116
24, 17.45	0.08	0.03	20.13	0.0015	79	1014
24, 20.10	0.07	0.03	20.13	0.0013	79	1014
25, 0.45	0.00	0.00	20.08	0.0000	79	850
25, 3.14	0.04	0.02	19.92	0.0016	80	875
25, 8.39	0.06	0.05	20.16	0.0013	79	1980
25, 12.29	0.08	0.04	19.94	0.0017	79	1008
25, 17.10	0.12	0.13	19.62	0.0020	80	1050
25, 20.10	0.10	0.10	19.85	0.0016	79	1020
26, 0.20	INSUFFICIENT SAMPLE TAKEN					1353
26, 3.25	0.12	0.13	19.48	0.0026	80	1716
26, 8.25	0.08	0.11	19.73	0.0021	80	1443
26, 12.09	0.05	0.09	19.78	0.0018	80	1482
26, 16.30	INSUFFICIENT SAMPLE TAKEN					1353
26, 20.10	0.08	0.09	19.97	0.0016	79	850
27, 0.20	0.08	0.07	19.98	0.0018	79	1023
27, 3.14	0.08	0.07	20.03	0.0016	79	858
27, 8.04	0.08	0.05	20.05	0.0016	79	957
27, 11.39	0.25	0.07	20.13	0.0017	79	726
27, 16.04	0.07	0.04	20.09	0.0014	79	792
28, 1.52	0.08	0.04	20.08	0.0017	79	720

NITROGEN BY DIFFERENCE

Table 5.10 Gas Samples at 95 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT,109 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.15	0.06	0.14	19.75	0.0024	80	1980
23, 17.34	0.06	0.13	19.63	0.0041	80	1530
23, 18.49	0.07	0.10	19.82	0.0030	80	1674
24, 1.15	INSUFFICIENT SAMPLE TAKEN					1450
24, 4.09	0.08	0.07	19.88	0.0030	79	1224
24, 9.55	0.08	0.05	20.07	0.0025	79	990
24, 11.39	0.09	0.05	20.00	0.0031	79	1296
24, 17.50	0.08	0.03	20.06	0.0022	79	1014
24, 20.15	0.07	0.03	20.07	0.0021	79	1014
25, 0.14	0.10	0.05	19.97	0.0026	79	816
25, 3.09	0.09	0.05	19.97	0.0028	79	816
25, 8.30	0.07	0.03	20.05	0.0023	79	1008
25, 12.34	0.06	0.03	19.98	0.0024	79	1116
25, 17.04	0.08	0.07	19.93	0.0024	79	960
25, 20.08	0.00	0.00	19.97	0.0025	80	1050
26, 0.08	INSUFFICIENT SAMPLE TAKEN					1353
26, 3.20	0.08	0.10	19.75	0.0031	80	1650
26, 8.19	0.07	0.08	19.73	0.0028	80	1482
26, 12.14	0.04	0.08	19.72	0.0027	80	1560
26, 16.30	0.10	0.08	19.79	0.0029	80	1054
26, 20.07	0.10	0.09	19.87	0.0035	79	816
27, 0.14	0.11	0.08	19.89	0.0028	79	1056
27, 3.07	INSUFFICIENT SAMPLE TAKEN					891
27, 7.59	0.32	0.11	19.93	0.0024	79	858
27, 11.44	0.09	0.04	20.06	0.0023	79	825
27, 16.02	0.08	0.04	19.93	0.0023	79	864
28, 1.55	0.08	0.05	19.89	0.0033	79	792

NITROGEN BY DIFFERENCE

Table 5.11 Gas Samples at 109 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT,125 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.15	0.53	0.59	15.58	0.0240	83	1100
23, 17.39	0.45	0.64	16.32	0.0210	82	1584
23, 18.49	0.42	0.61	15.59	0.0275	83	2160
24, 1.21	0.39	0.37	18.14	0.0110	81	1368
24, 4.20	0.40	0.37	17.53	0.0160	81	1350
24, 10.04	0.26	0.38	18.19	0.0130	81	1116
24, 11.45	0.21	0.35	18.72	0.0110	80	1134
24, 17.54	0.31	0.20	18.62	0.0110	80	1034
24, 20.16	0.25	0.15	19.14	0.0069	80	1092
25, 0.04	0.28	0.22	18.29	0.0009	81	850
25, 3.05	0.36	0.26	17.82	0.0013	81	924
25, 8.25	0.35	0.32	17.85	0.0160	81	1332
25, 12.45	0.38	0.35	17.43	0.0180	81	1152
25, 16.59	0.20	0.25	18.28	0.0110	81	990
25, 20.04	0.31	0.29	18.27	0.0130	81	1050
26, 0.04	INSUFFICIENT SAMPLE TAKEN					1320
26, 3.09	0.38	0.44	17.71	0.0120	81	1440
26, 8.15	0.32	0.45	17.63	0.0130	81	1521
26, 12.25	0.32	0.48	17.23	0.0170	81	1462
26, 16.20	0.32	0.33	18.02	0.0130	81	1190
26, 20.04	0.36	0.40	17.29	0.0180	81	1020
27, 0.09	0.34	0.33	17.82	0.0011	81	1056
27, 3.02	0.34	0.30	18.02	0.0010	81	660
27, 7.55	0.34	0.28	18.01	0.0160	81	990
27, 11.50	0.34	0.27	17.92	0.0173	81	825
27, 16.00	0.34	0.27	18.09	0.0140	81	900
28, 1.59	0.46	0.36	16.51	0.0270	82	829

NITROGEN BY DIFFERENCE

Table 5.12 Gas Samples at 125 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT,141 CHOCK

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.19	0.00	0.00	0.09	0.0000	99	2200
23, 17.45	INSUFFICIENT SAMPLE TAKEN					1600
23, 18.55	0.07	0.15	19.89	0.0012	79	1512
24, 1.30	0.09	0.12	19.85	0.0018	79	1278
24, 4.24	0.09	0.12	19.93	0.0013	79	1190
24, 10.09	INSUFFICIENT SAMPLE TAKEN					1314
24, 11.50	0.20	0.35	19.62	0.0016	79	1135
24, 18.00	0.23	0.47	19.31	0.0017	79	1014
24, 20.19	INSUFFICIENT SAMPLE TAKEN					1014
24, 23.54	0.14	0.19	19.81	0.0013	79	858
25, 2.56	INSUFFICIENT SAMPLE TAKEN					884
25, 8.19	0.16	0.21	19.68	0.0019	79	986
25, 12.29	INSUFFICIENT SAMPLE TAKEN					1296
25, 16.55	INSUFFICIENT SAMPLE TAKEN					1200
25, 19.59	0.15	0.20	19.65	0.0012	79	1080
26, 0.00	0.11	0.17	19.79	0.0012	79	1815
26, 3.05	0.11	0.15	19.66	0.0015	80	1533
26, 8.10	0.06	0.20	19.57	0.0011	80	1560
26, 12.29	0.07	0.22	19.51	0.0012	80	1482
26, 16.18	0.20	0.42	19.15	0.0014	80	1156
26, 19.59	INSUFFICIENT SAMPLE TAKEN					1020
27, 0.04	0.20	0.33	19.47	0.0013	79	1056
27, 3.00	0.15	0.24	19.69	0.0013	79	825
27, 7.50	0.16	0.28	19.70	0.0013	79	792
27, 11.56	0.34	0.27	19.57	0.0060	79	759
27, 15.55	0.21	0.44	18.31	0.0013	81	864
28, 2.04	0.09	0.10	19.97	0.0010	79	864

NITROGEN BY DIFFERENCE

Table 5.13 Gas Samples at 141 Chock

GAS ANALYSIS 23/1/84 TO 28/1/84
SAMPLE POINT,10 YDS OB TAIL GATE RIP

DAY,TIME	CO2%	CH4%	O2%	CO%	N2%	SF6*10^12
23, 13.19	0.10	0.20	19.71	0.0011	79	2420
23, 16.30	0.08	0.19	19.87	0.0004	79	1638
24, 1.39	0.12	0.11	20.03	0.0005	79	1320
24, 8.10	0.07	0.07	20.23	0.0005	79	1206
24, 16.24	0.09	0.05	20.17	0.0008	79	1014
24, 23.39	0.07	0.06	20.14	0.0006	79	816
25, 8.14	0.07	0.04	20.21	0.0004	79	1008
25, 16.50	0.07	0.07	20.04	0.0004	79	1110
25, 23.50	0.08	0.10	19.98	0.0006	79	1485
26, 8.10	0.08	0.14	19.74	0.0005	80	1560
26, 16.15	0.08	0.13	19.93	0.0005	79	1152
27, 0.00	0.09	0.10	19.98	0.0010	79	1188
27, 12.20	0.12	0.11	20.02	0.0006	79	858
27, 15.49	0.08	0.06	20.14	0.0013	79	864
28, 2.10	0.07	0.05	20.18	0.0006	79	792

NITROGEN BY DIFFERENCE

Table 5.14 Gas Samples at the 10m Return Rip Sample Point

SPECIAL GENERAL BODY GAS SAMPLES 26/1/84

POSITION IN INTAKE	CO2 %	CH4 %	O2 %	N2 %	CO %	SF6*10^12
1020M	0.05	0.02	20.83	79.09	0.0001	90
1050M	0.06	0.02	20.74	79.18	0.0005	270
1060M	0.06	0.03	20.62	79.29	0.0003	1290
1090M	0.06	0.02	20.57	79.35	0.0003	1400
1120M	0.07	0.02	20.52	79.39	0.0003	1314
1140M	0.05	0.02	20.49	79.44	0.0002	1326
1170M	0.07	0.03	20.44	79.46	0.0003	1296
1240/ SWITCHGEARS		INSUFFICIENT SAMPLE				1800
INTAKE RIP	0.07	0.02	20.42	79.49	0.0003	1800

Table 5.15 Gas Samples Taken Along Intake Roadway

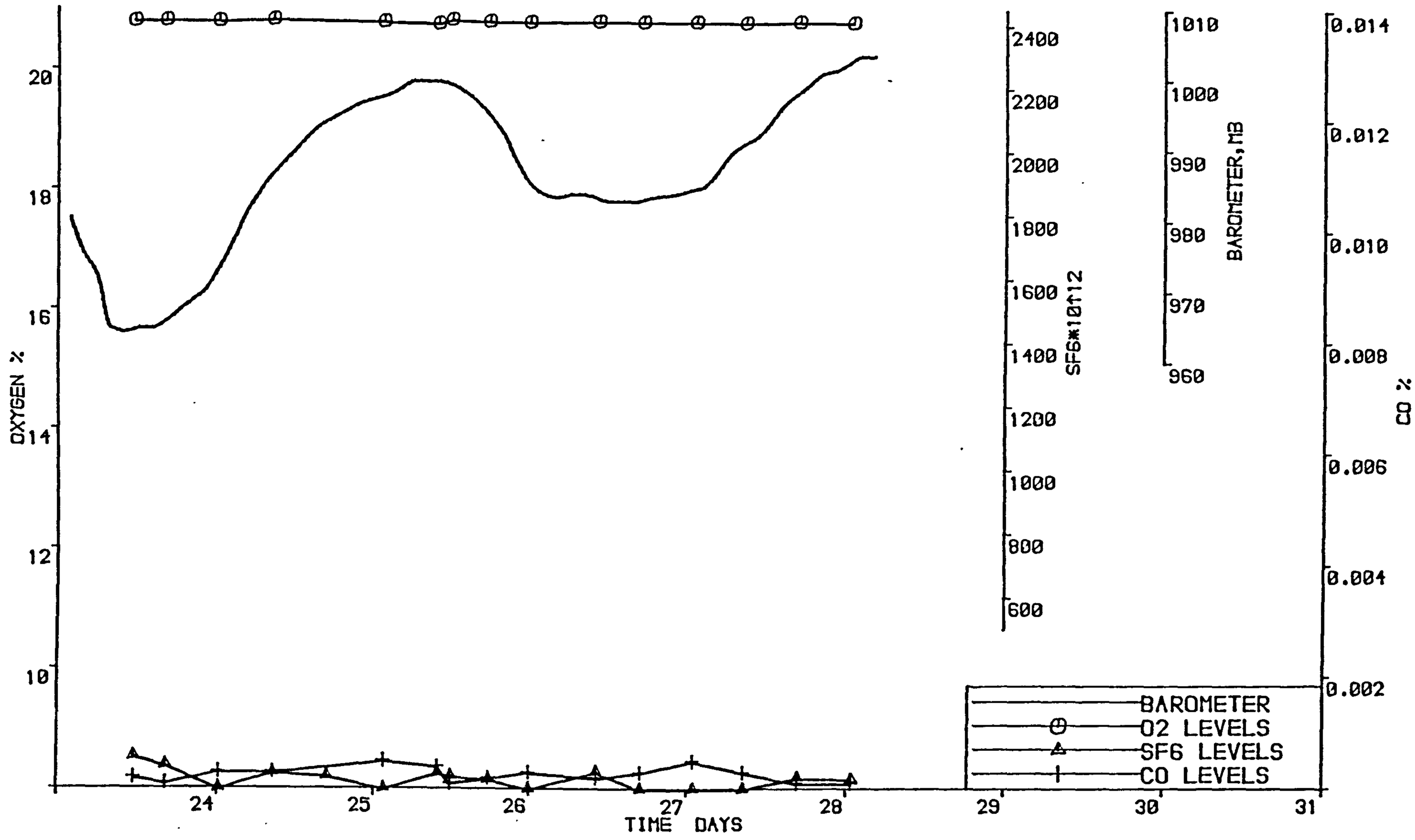


FIGURE 5.6 940m INTAKE SAMPLES

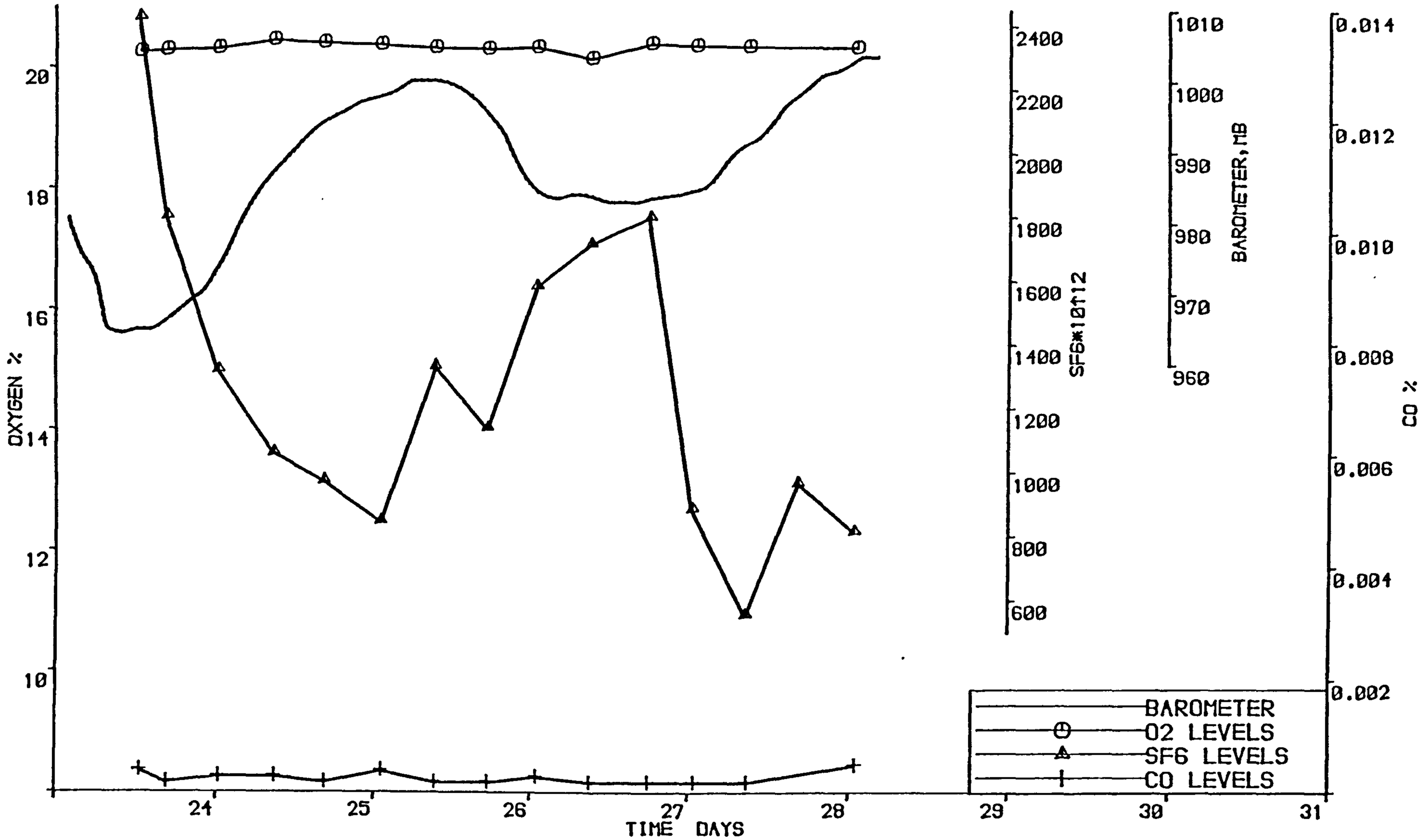


FIGURE 5.7 INTAKE RIP SAMPLES

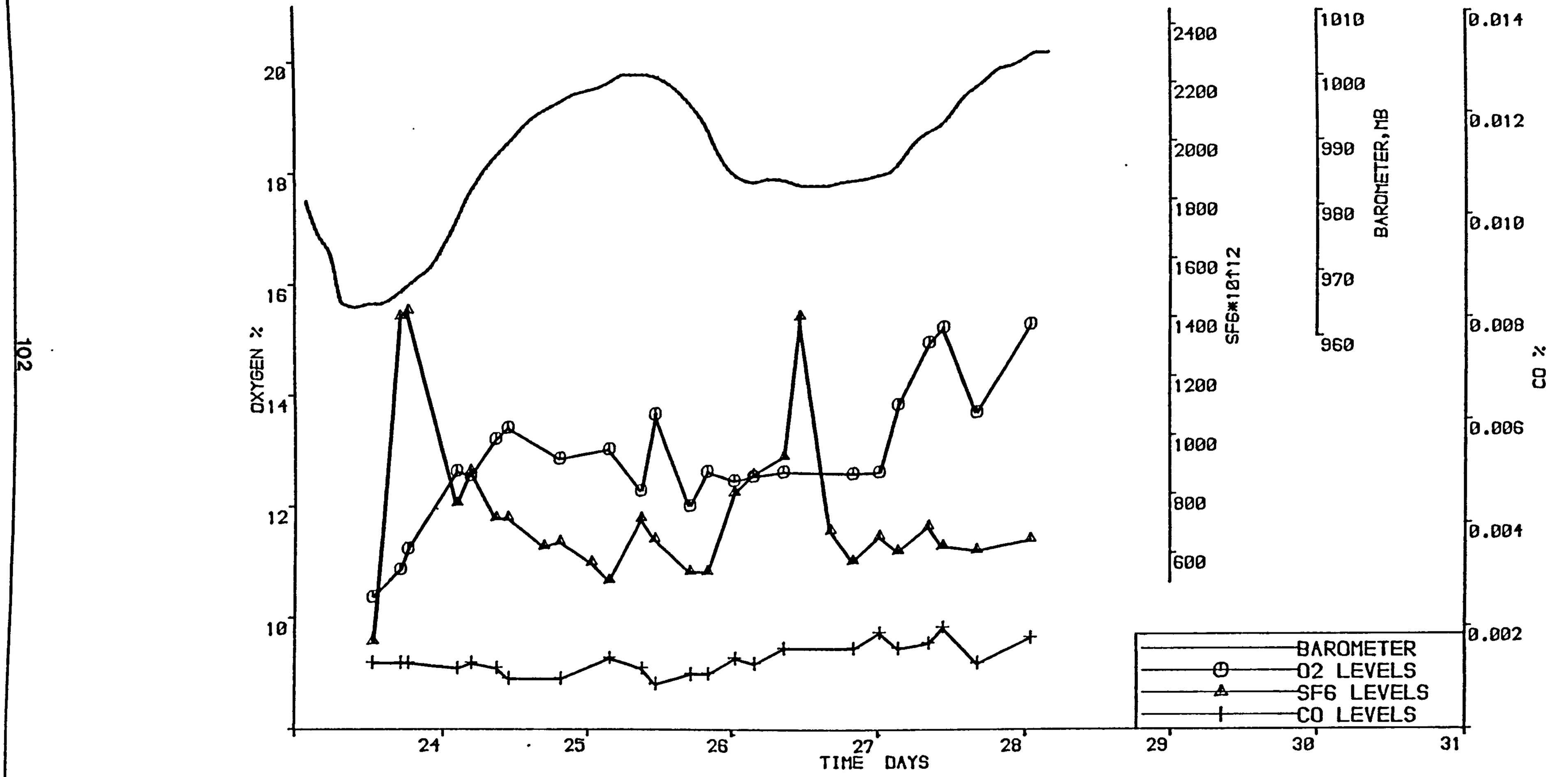


FIGURE 5.8 CHOCK 6 SAMPLES

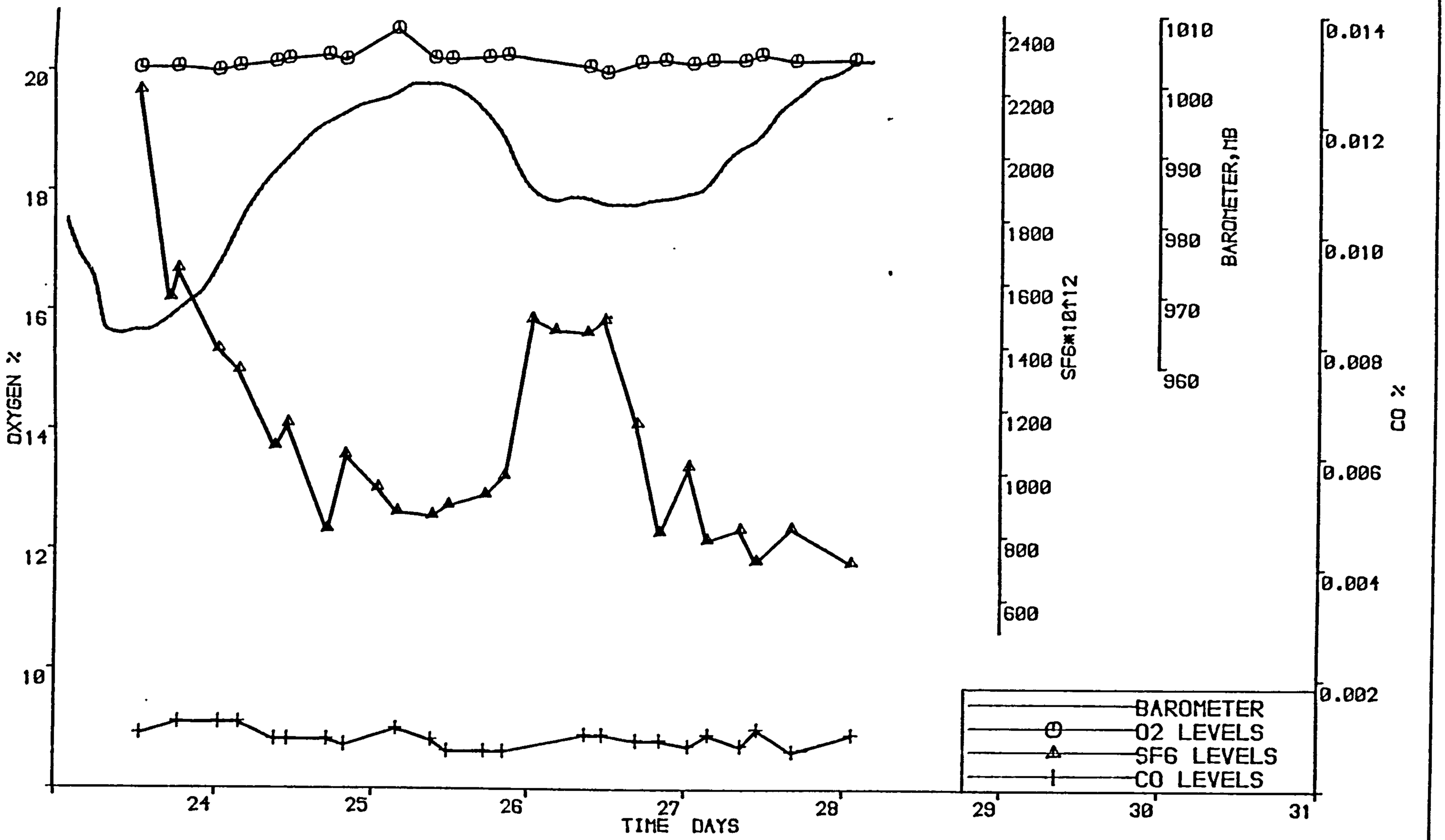


FIGURE 5.9 CHOCK 19 SAMPLES

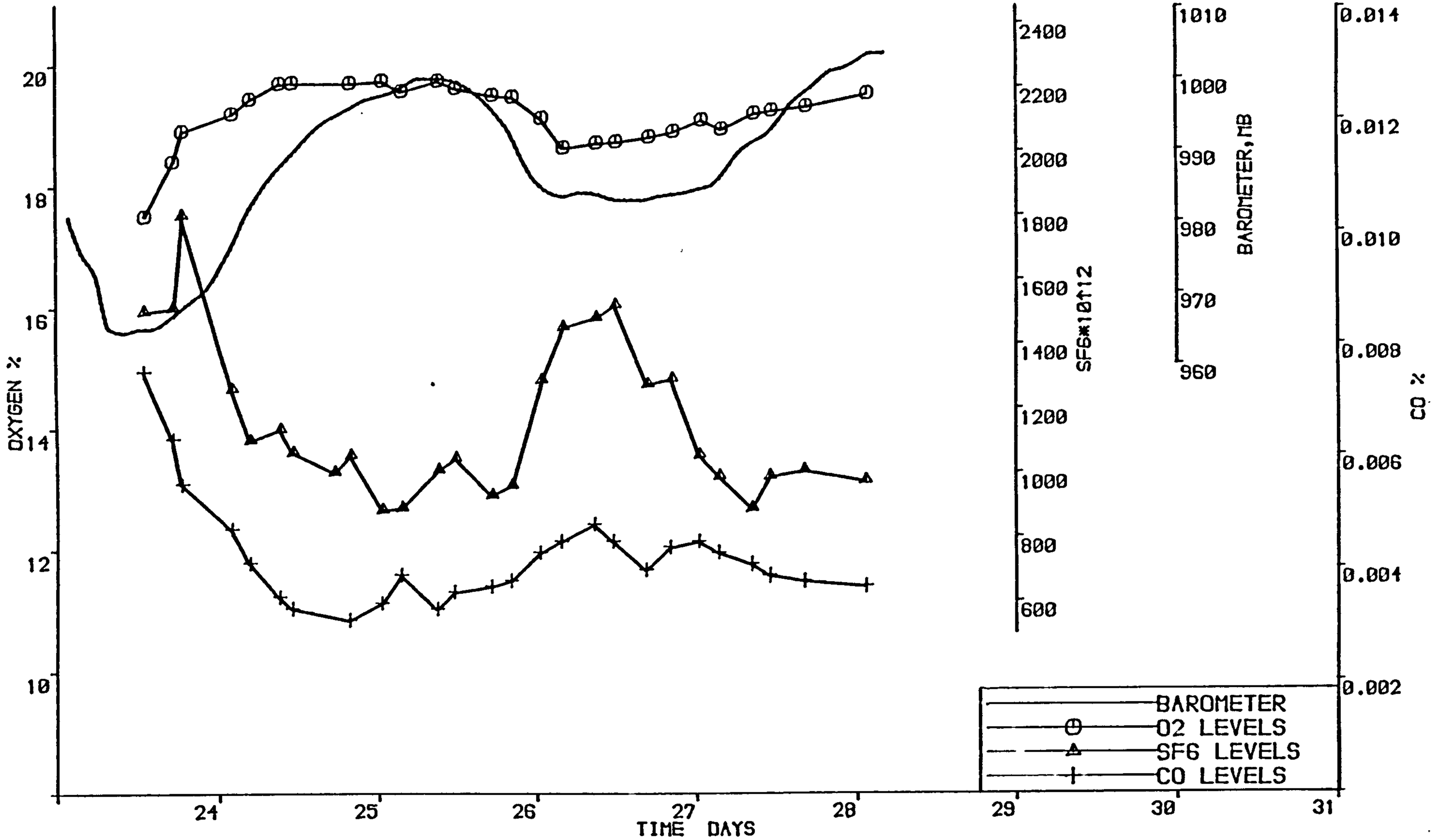


FIGURE 5.10 CHOCK 35 SAMPLES

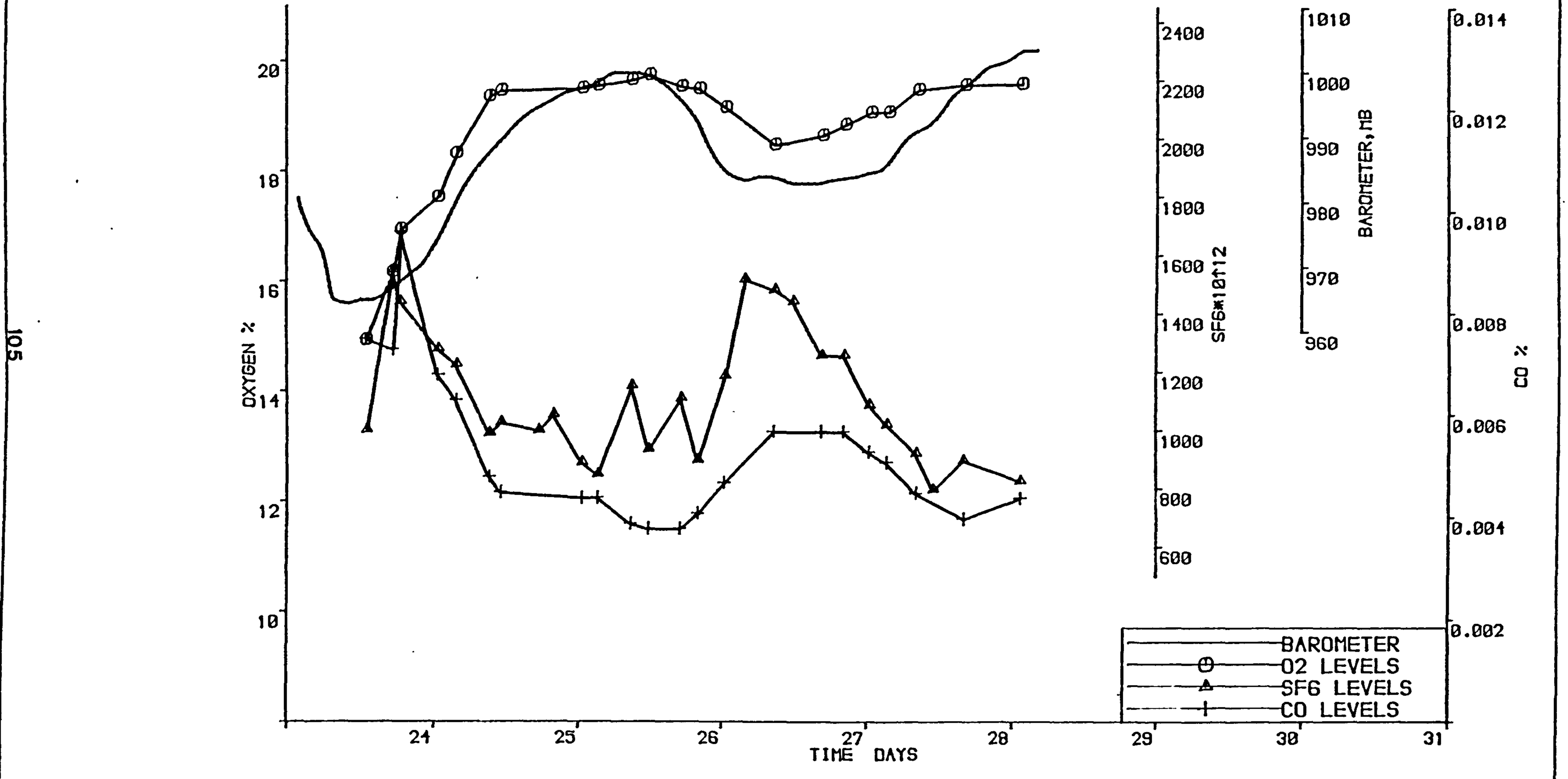


FIGURE 5.11 CHOCK 48 SAMPLES

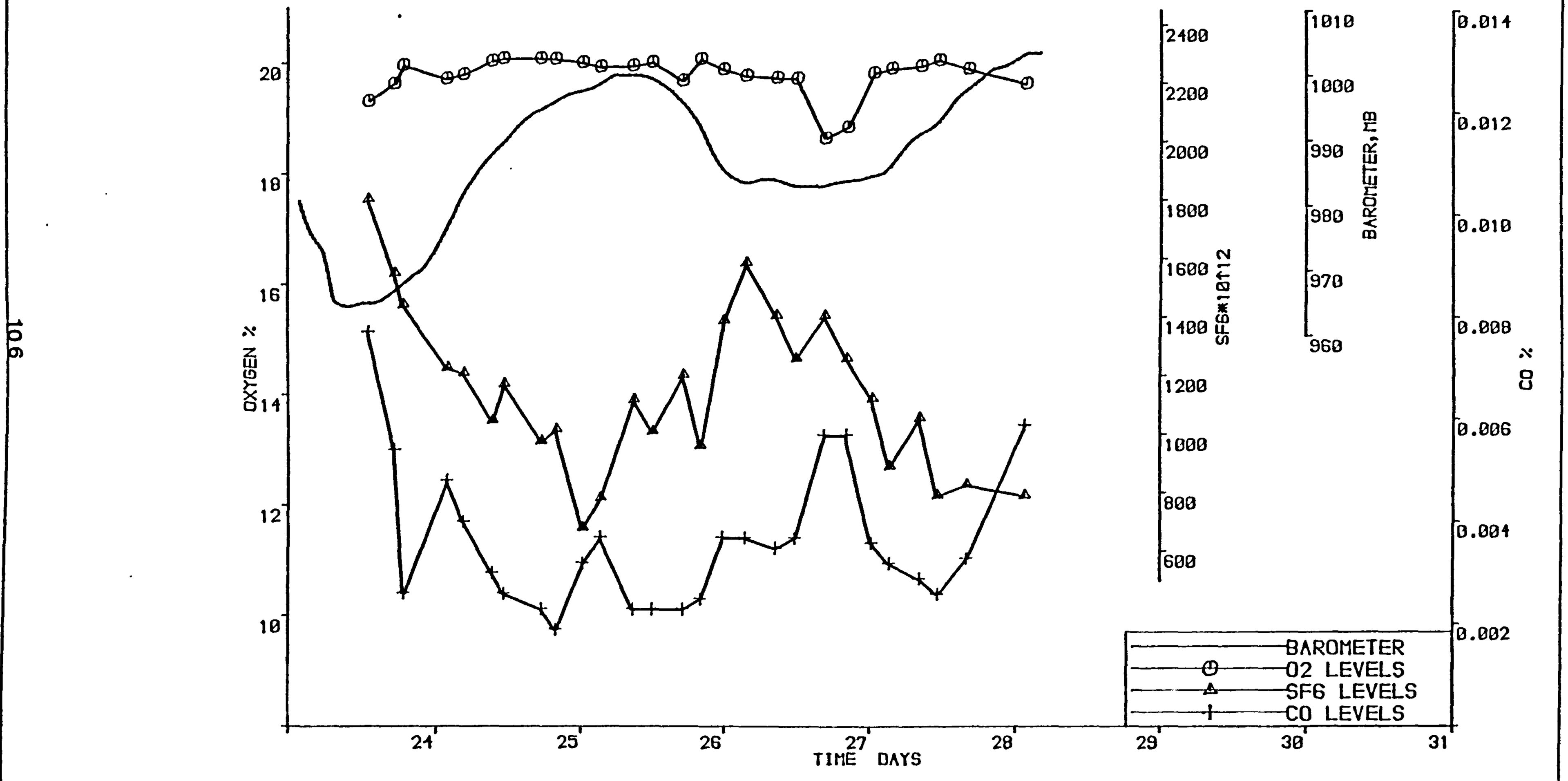


FIGURE 5.12 CHOCK 57 SAMPLES

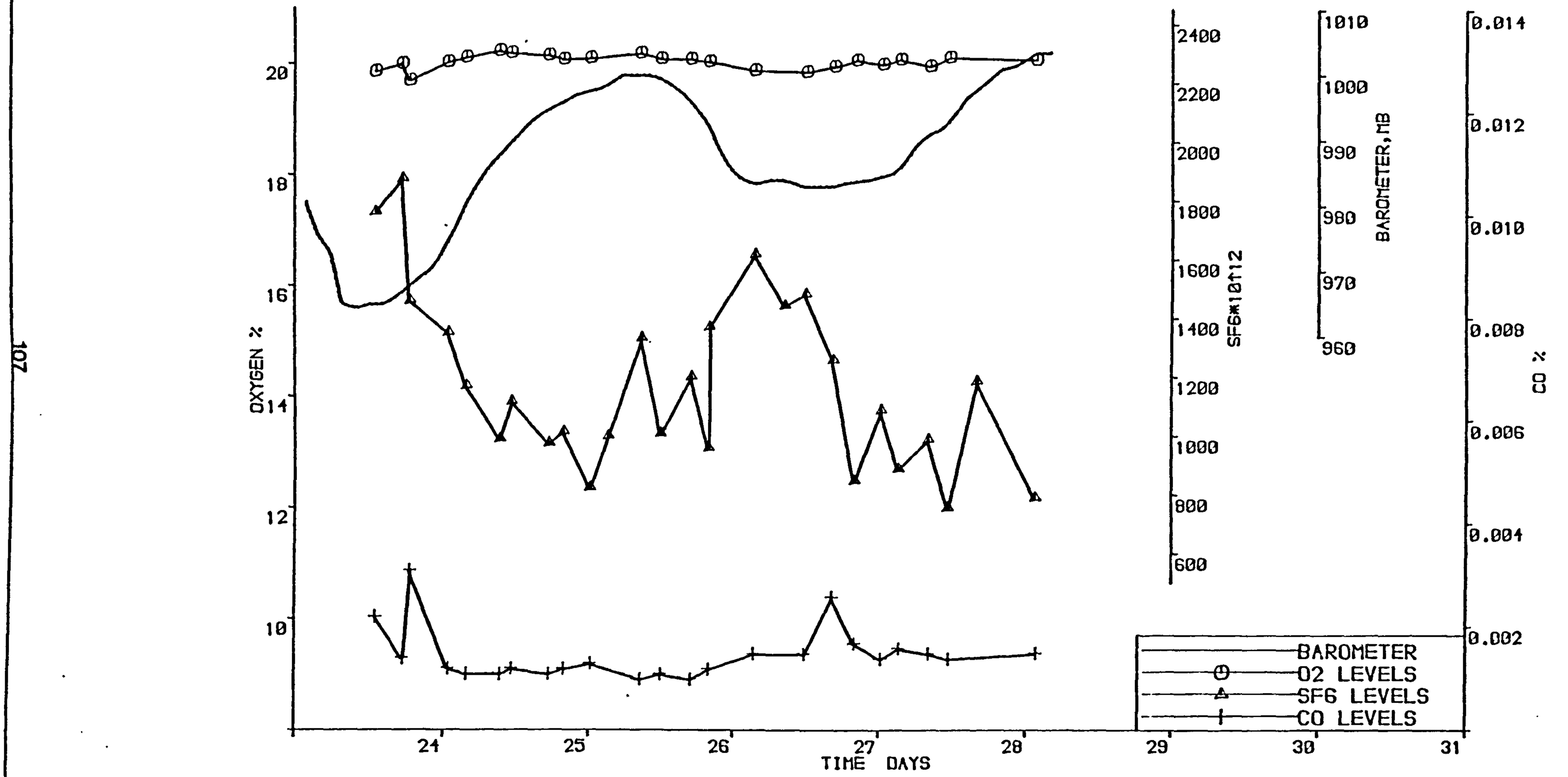


FIGURE 5.13 CHOCK 70 SAMPLES

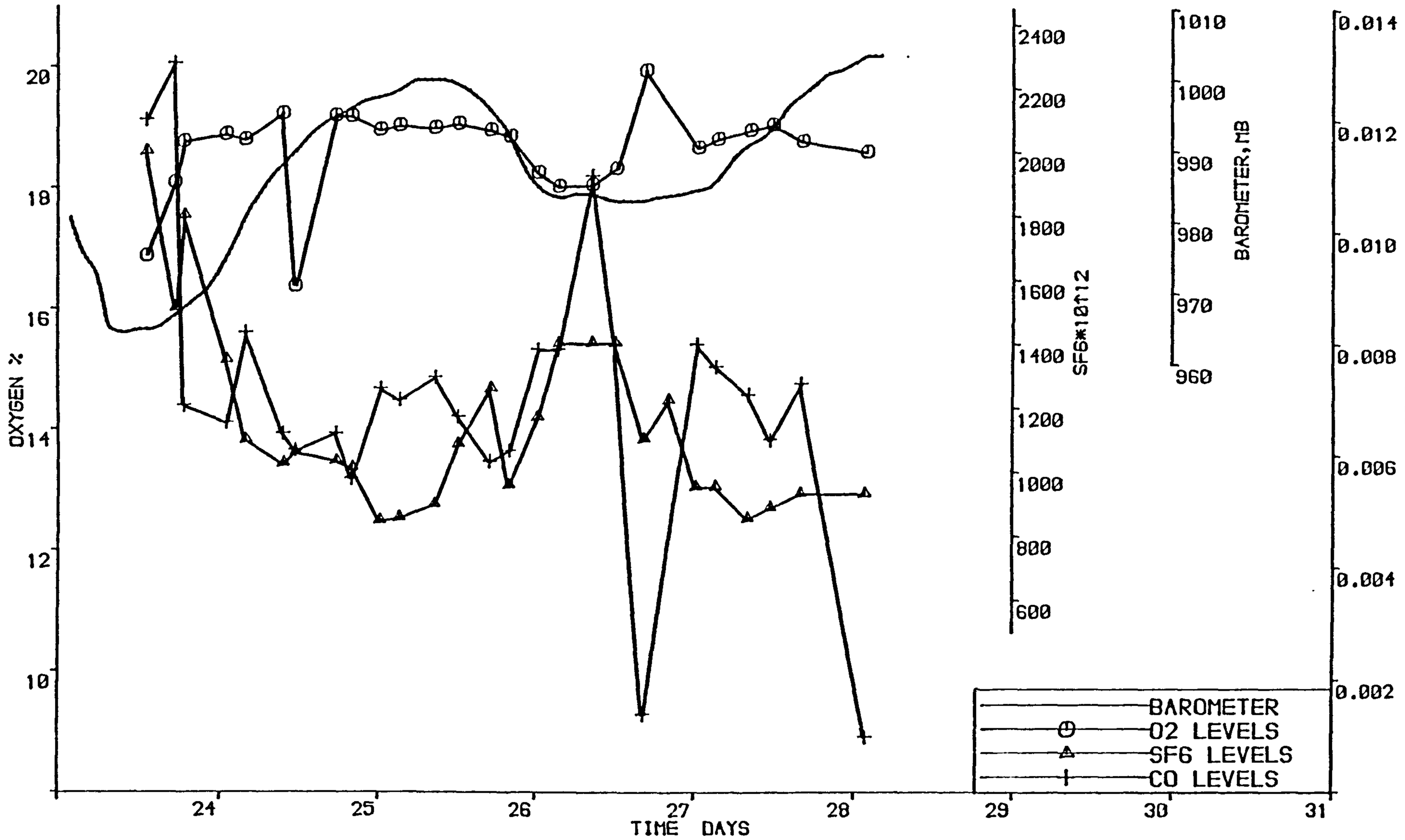


FIGURE 5.14 CHOCK 85 SAMPLES

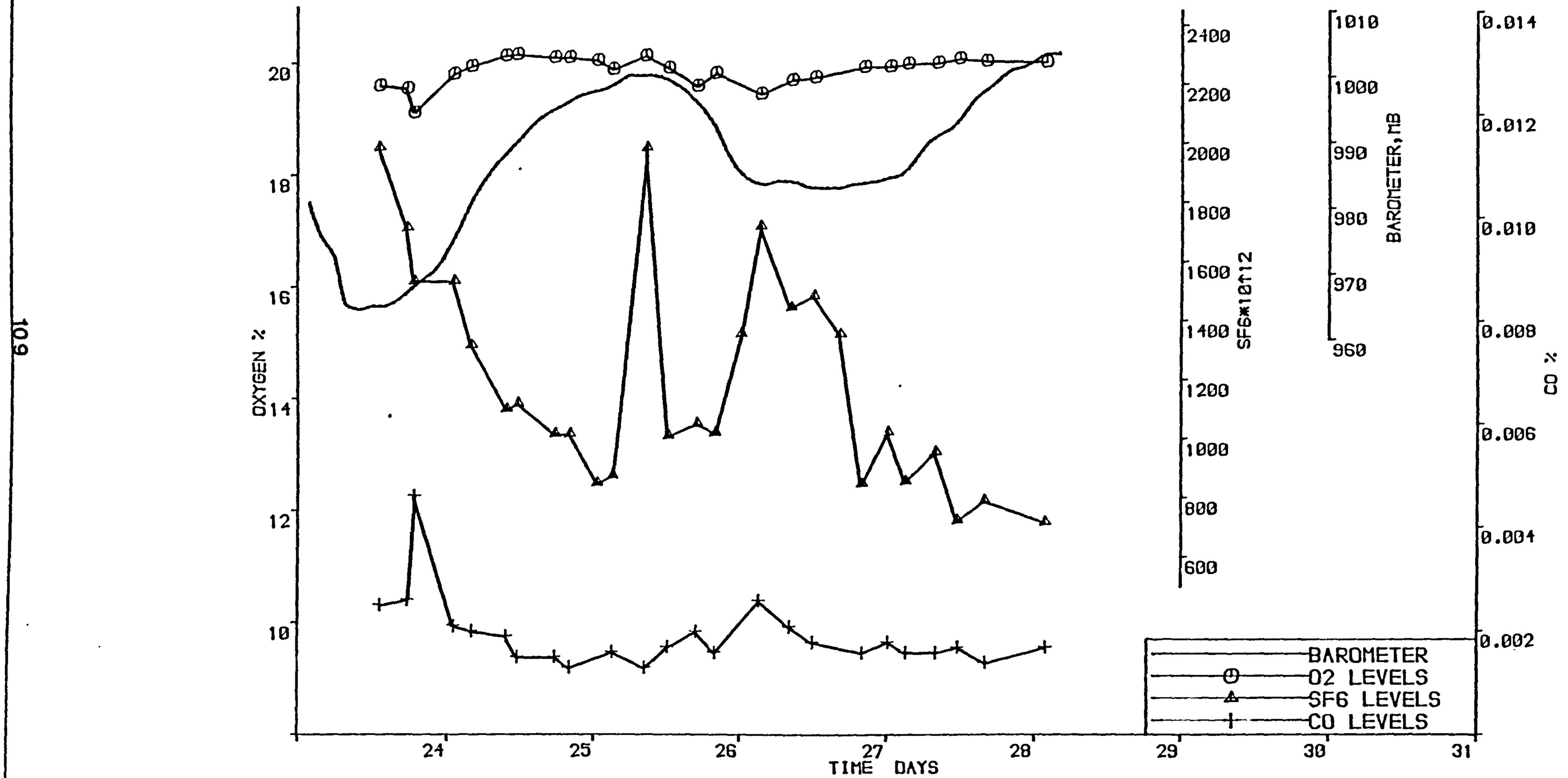


FIGURE 5.15 CHOCK 95 SAMPLES

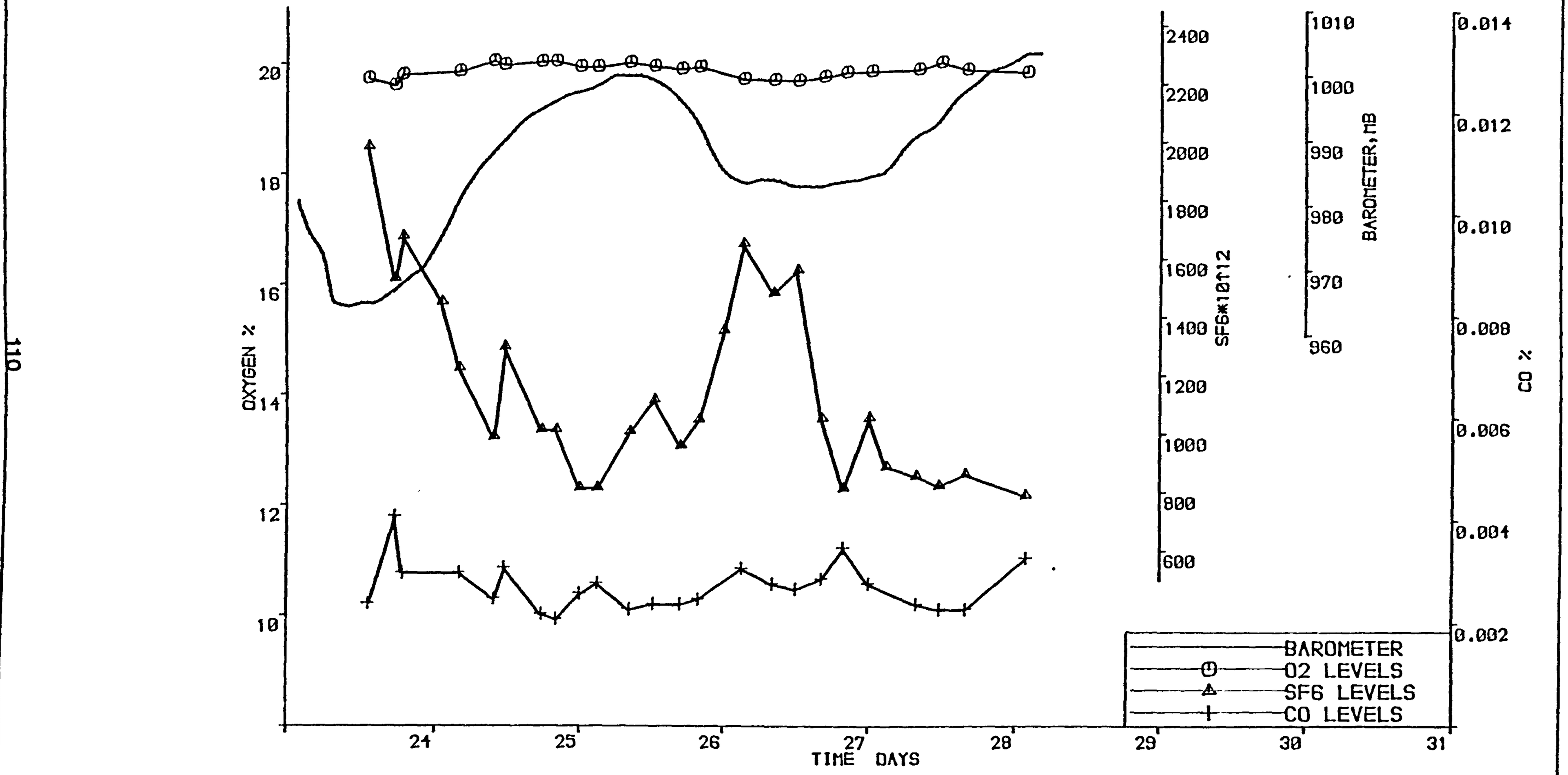


FIGURE 5.16 CHOCK 109 SAMPLES

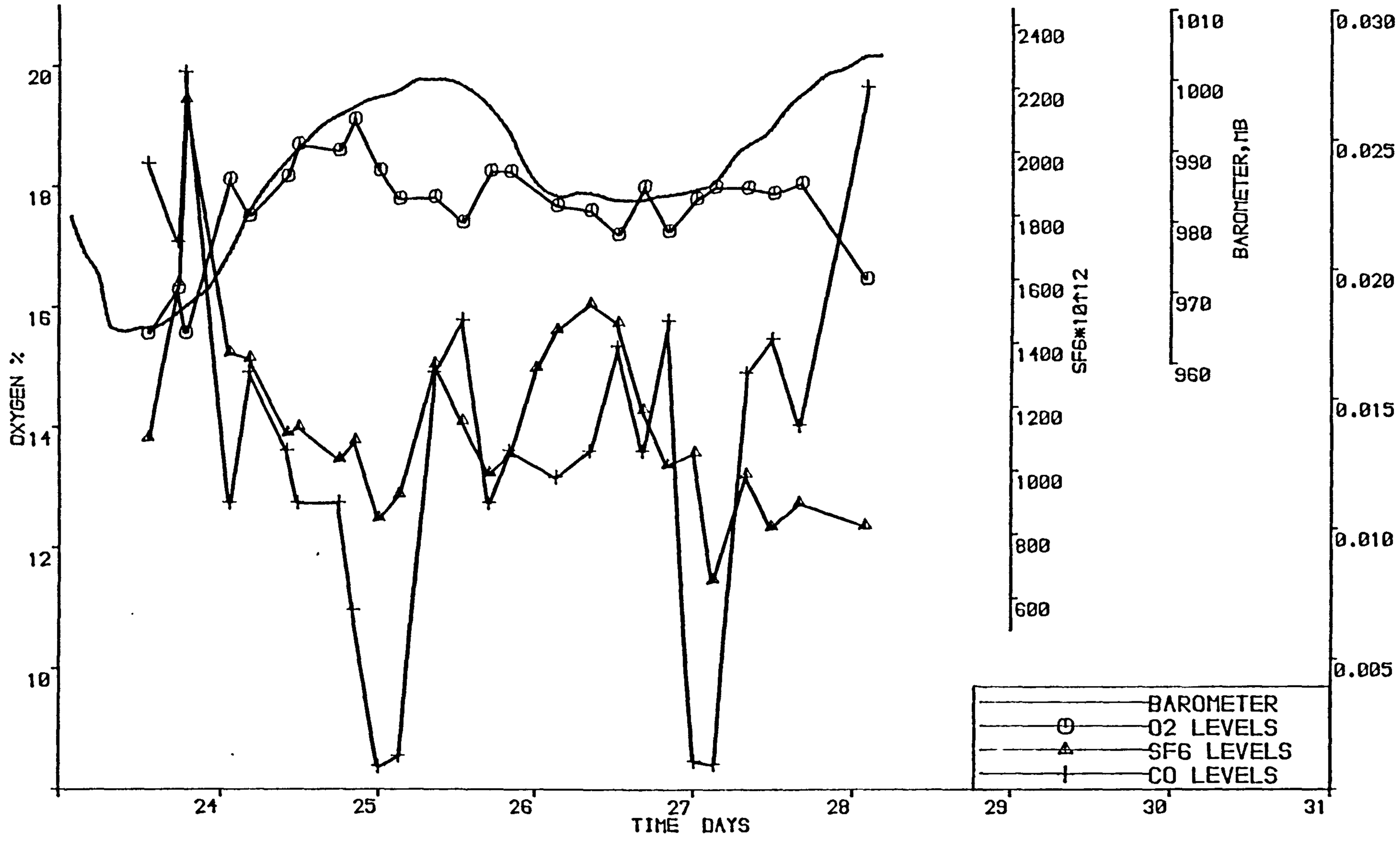


FIGURE 5.17 CHOCK 125 SAMPLES

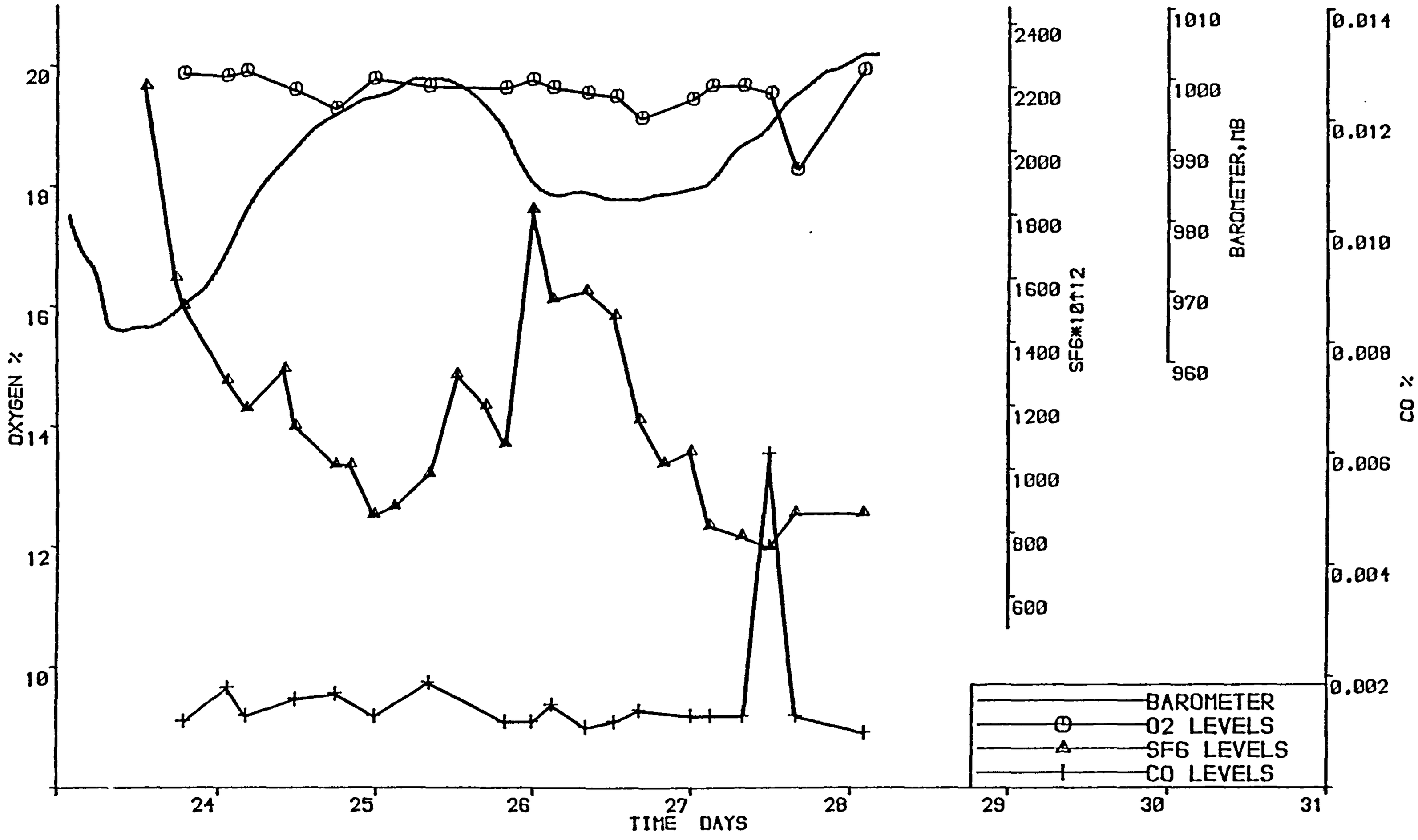


FIGURE 5.18 CHOCK 141 SAMPLES

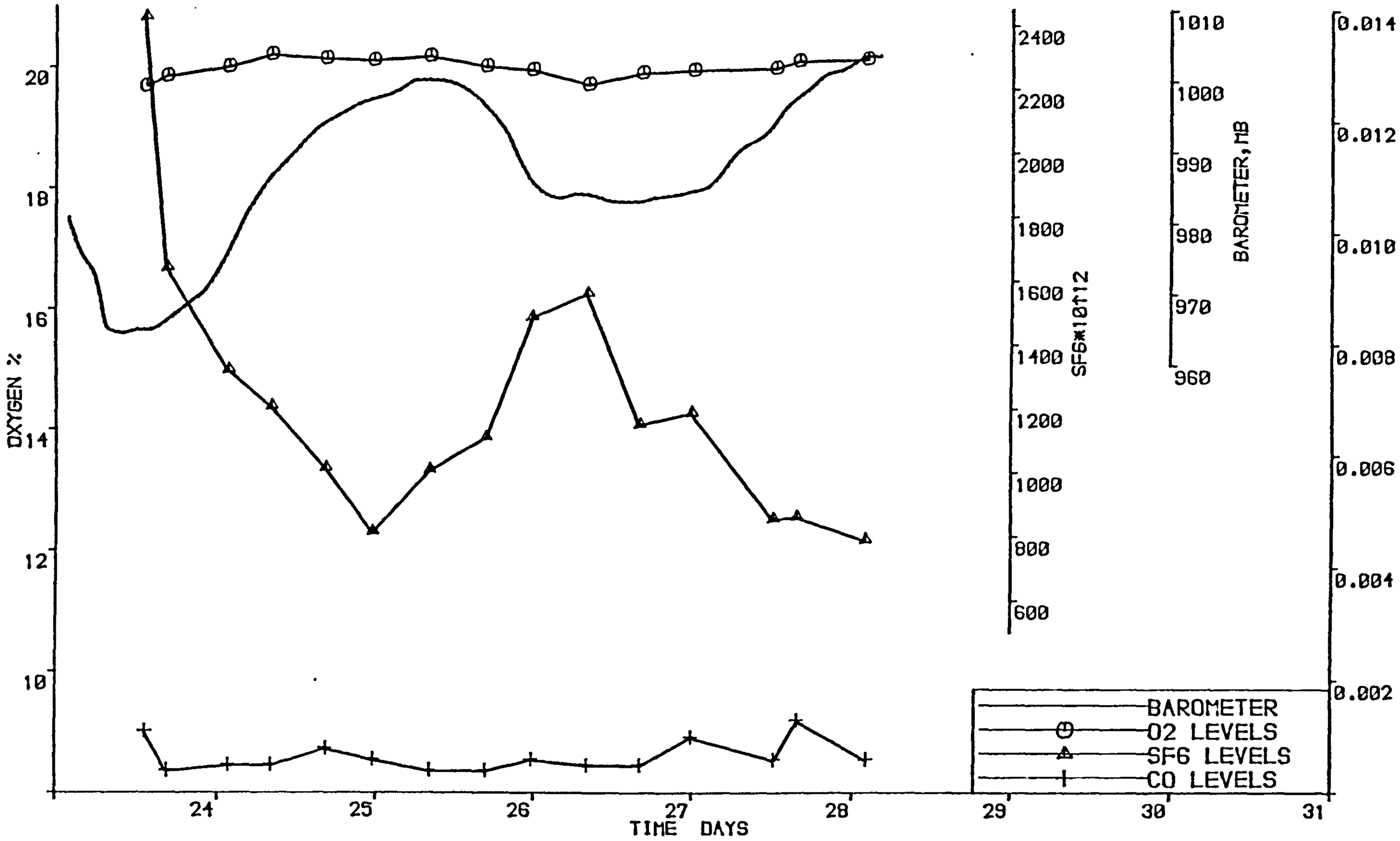


FIGURE 5.19 RETURN RIP SAMPLES

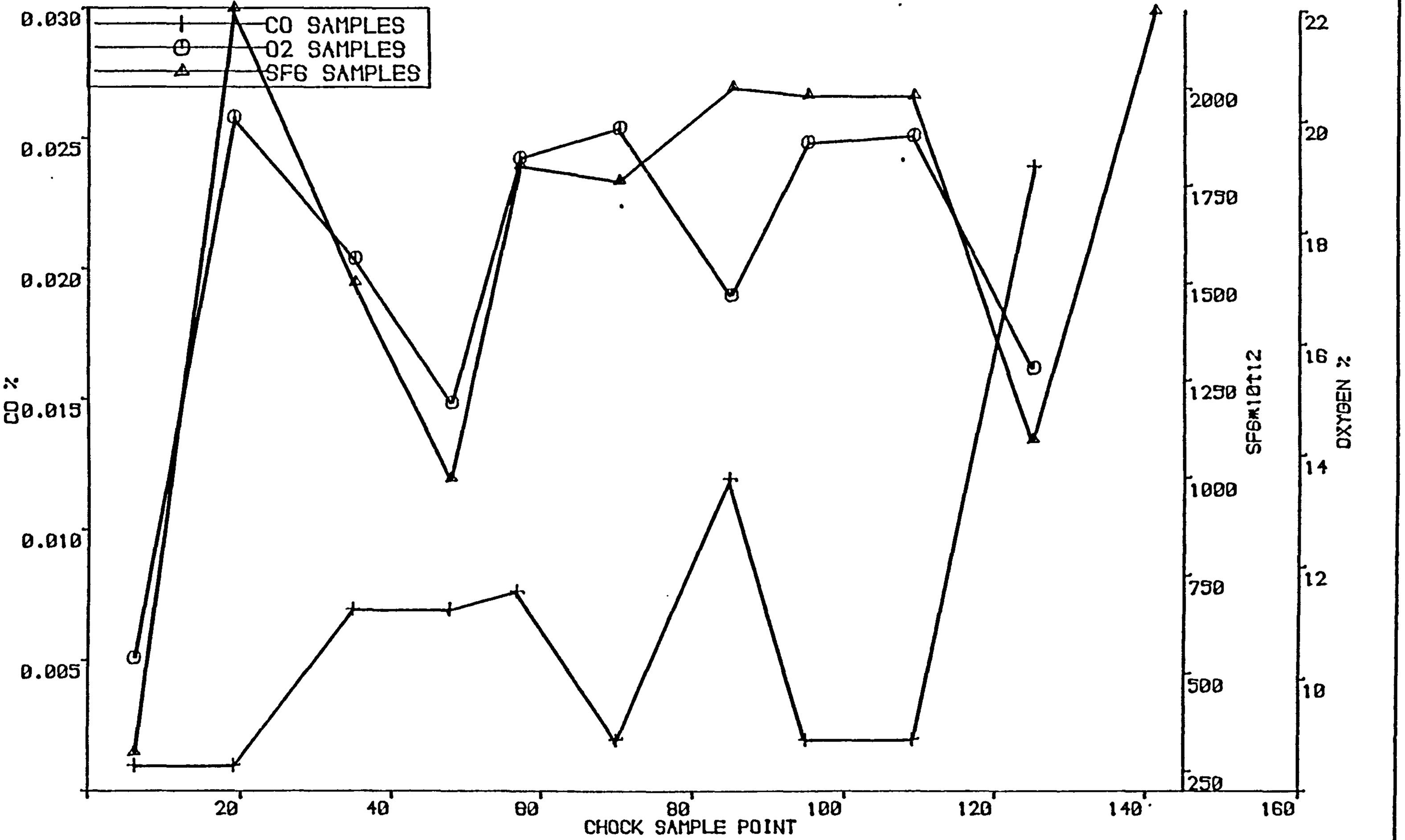


FIGURE 5.20 SAMPLES AT 12.50 ON 23/1

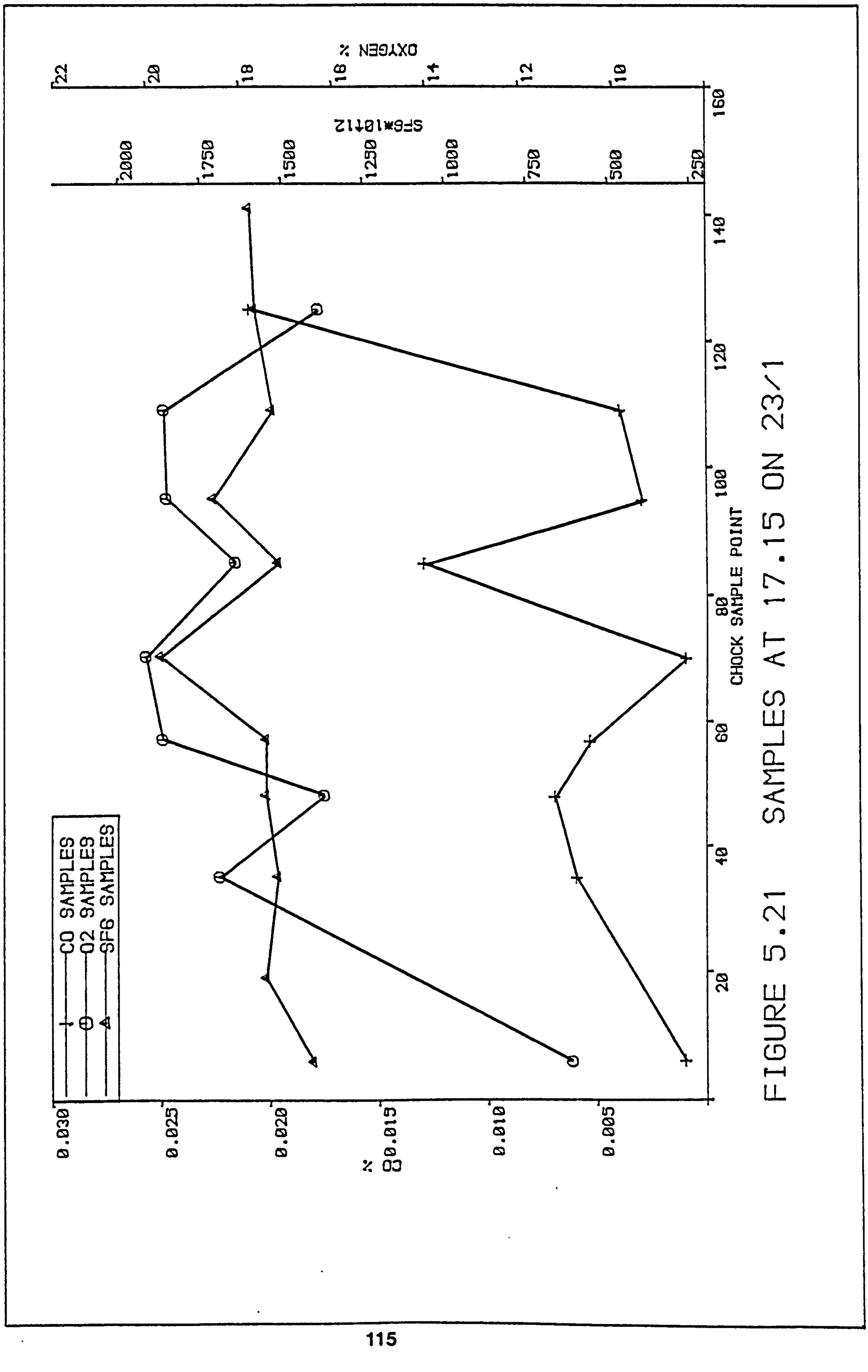


FIGURE 5.21 SAMPLES AT 17.15 ON 23/1

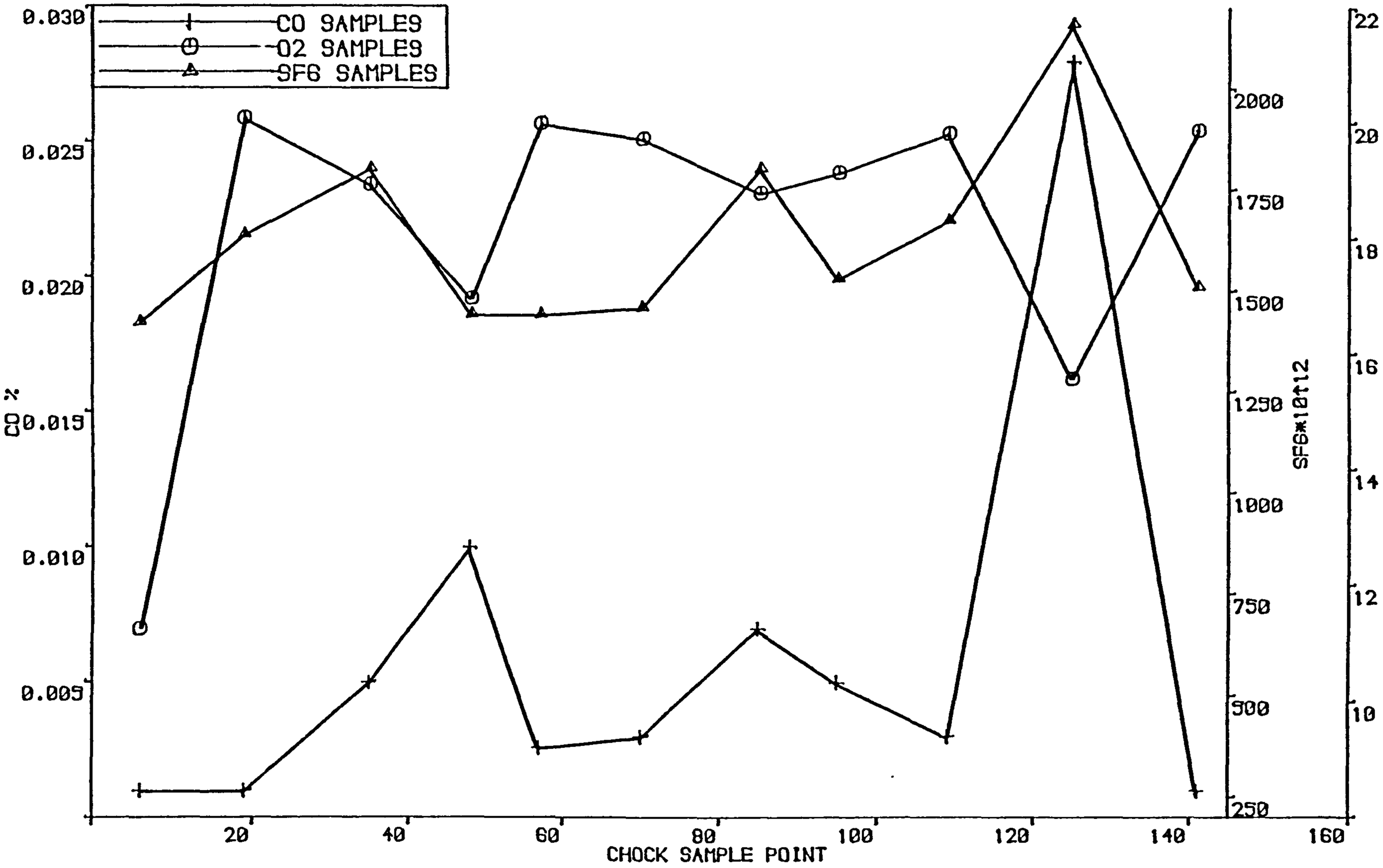


FIGURE 5.22 SAMPLES AT 18.30 ON 23/1

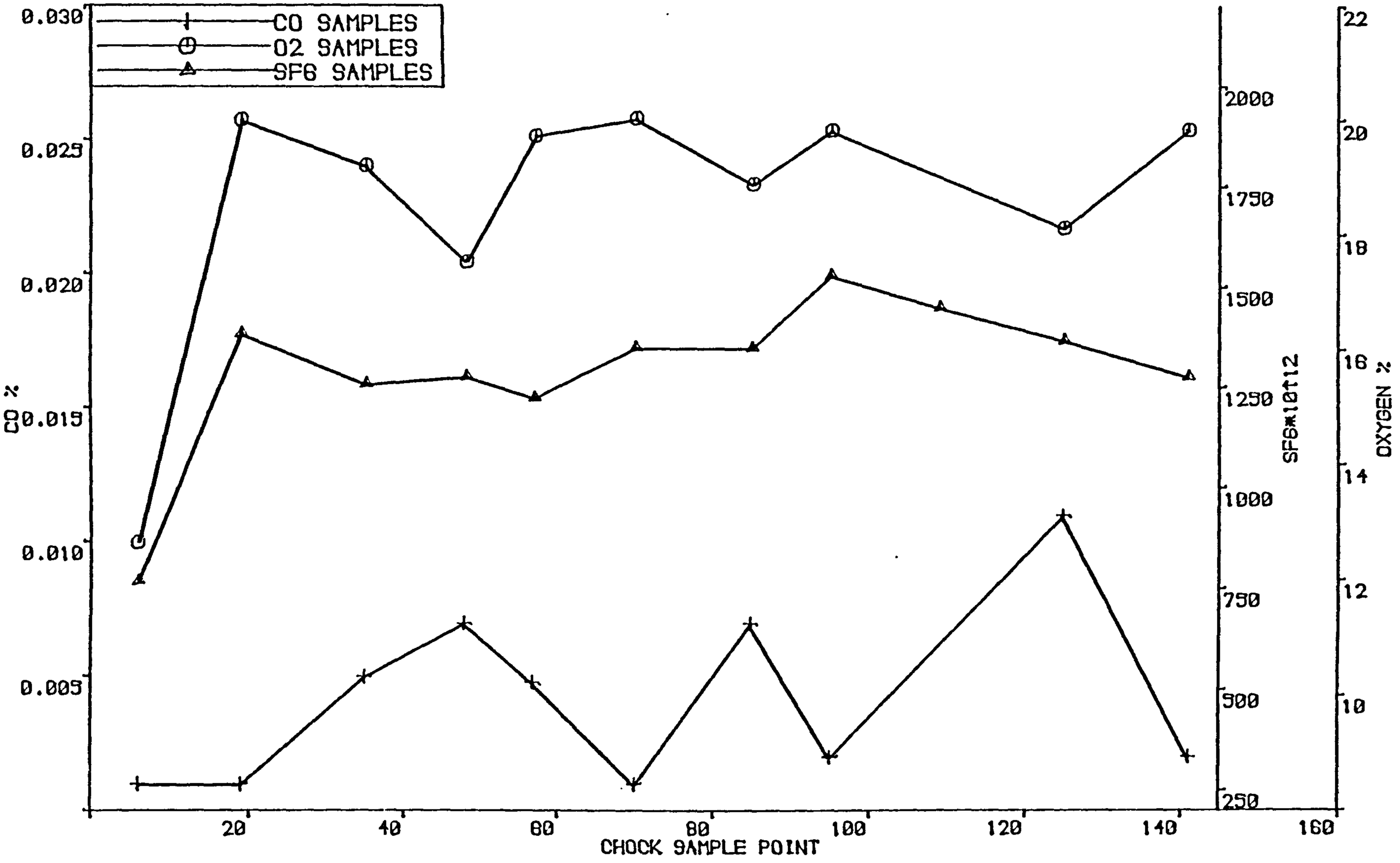


FIGURE 5.23 SAMPLES AT 00.30 ON 24/1

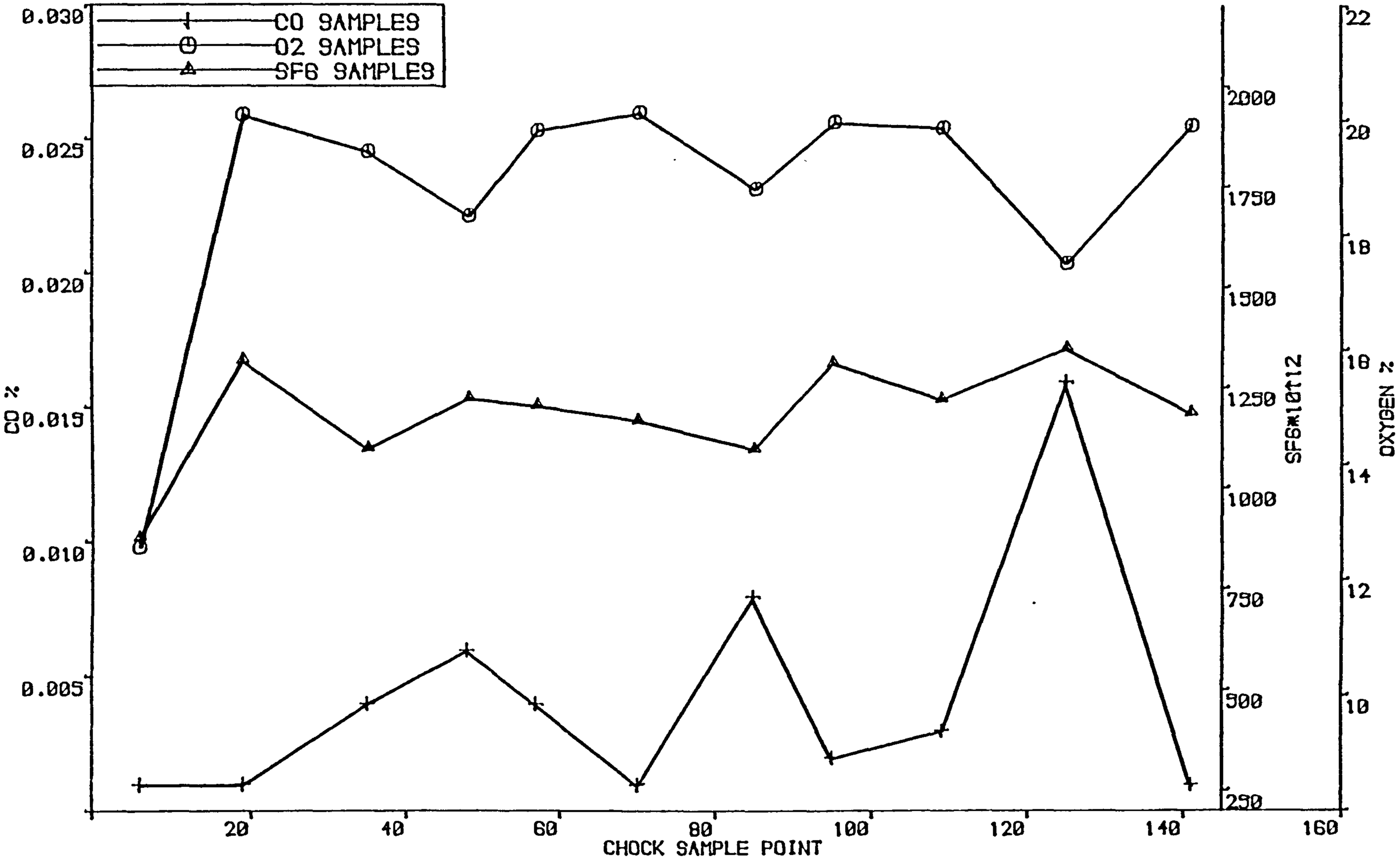


FIGURE 5.24 SAMPLES AT 03.30 ON 24/1

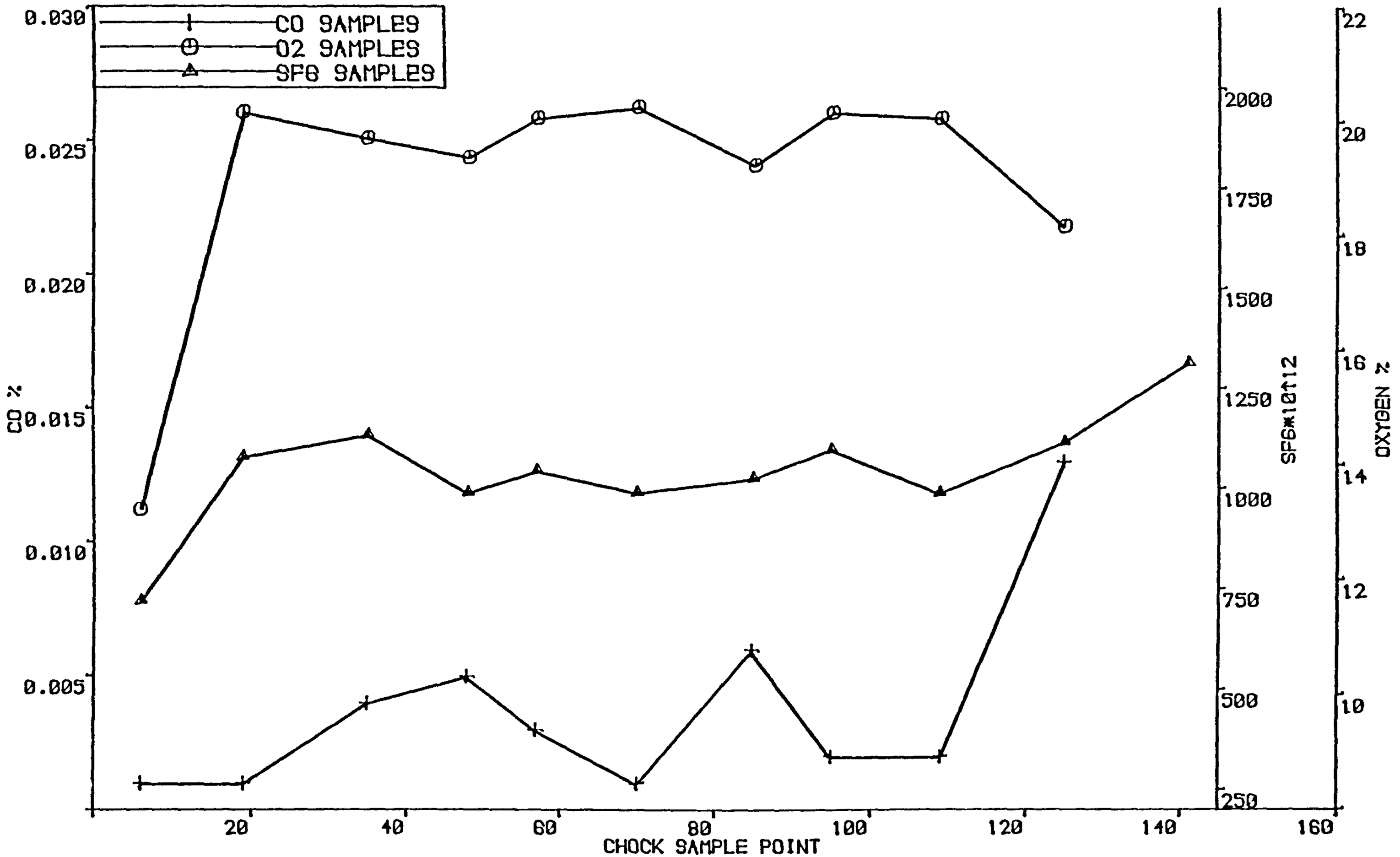


FIGURE 5.25 SAMPLES AT 09.10 ON 24/1

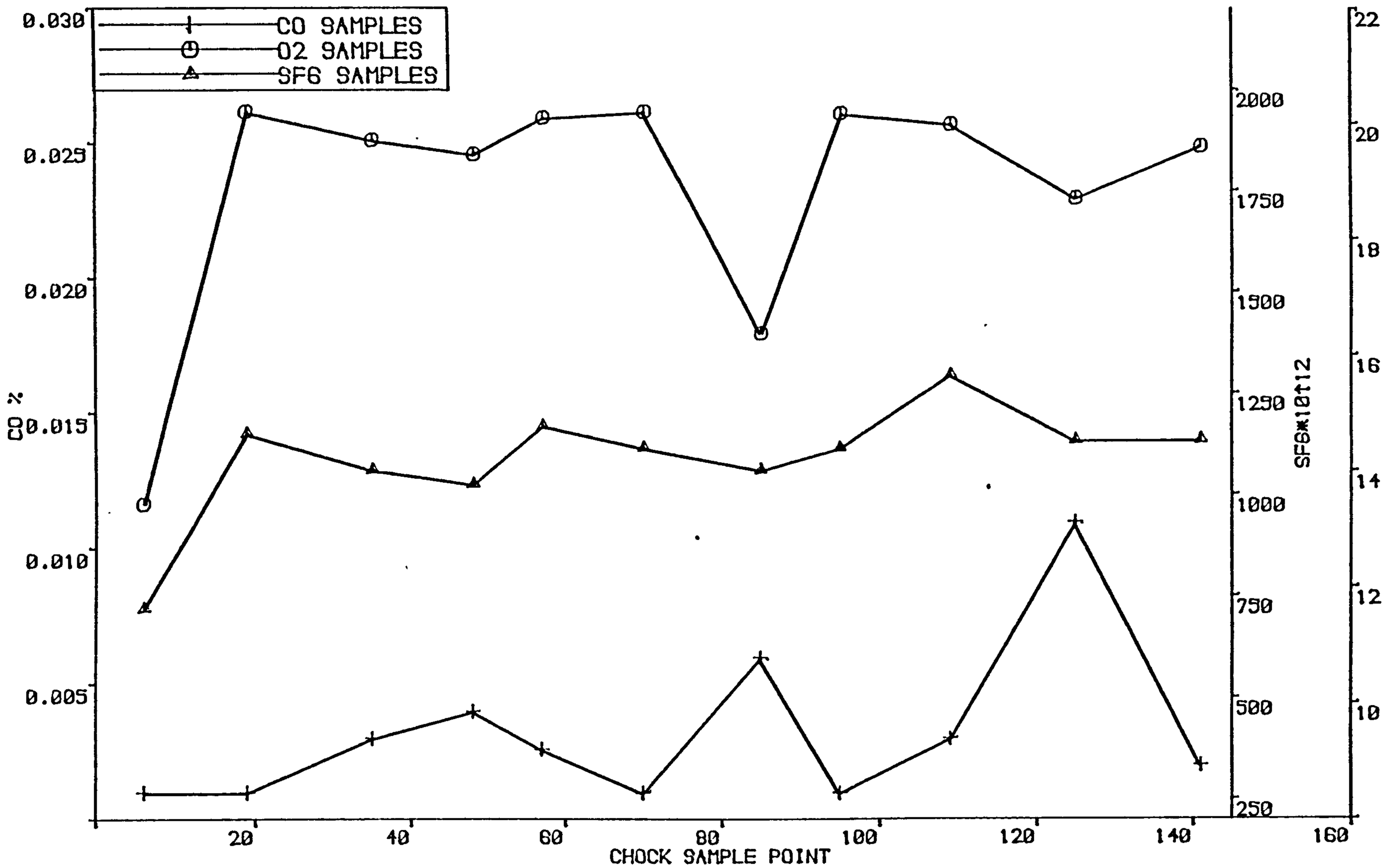


FIGURE 5.26 SAMPLES AT 11.05 ON 24/1

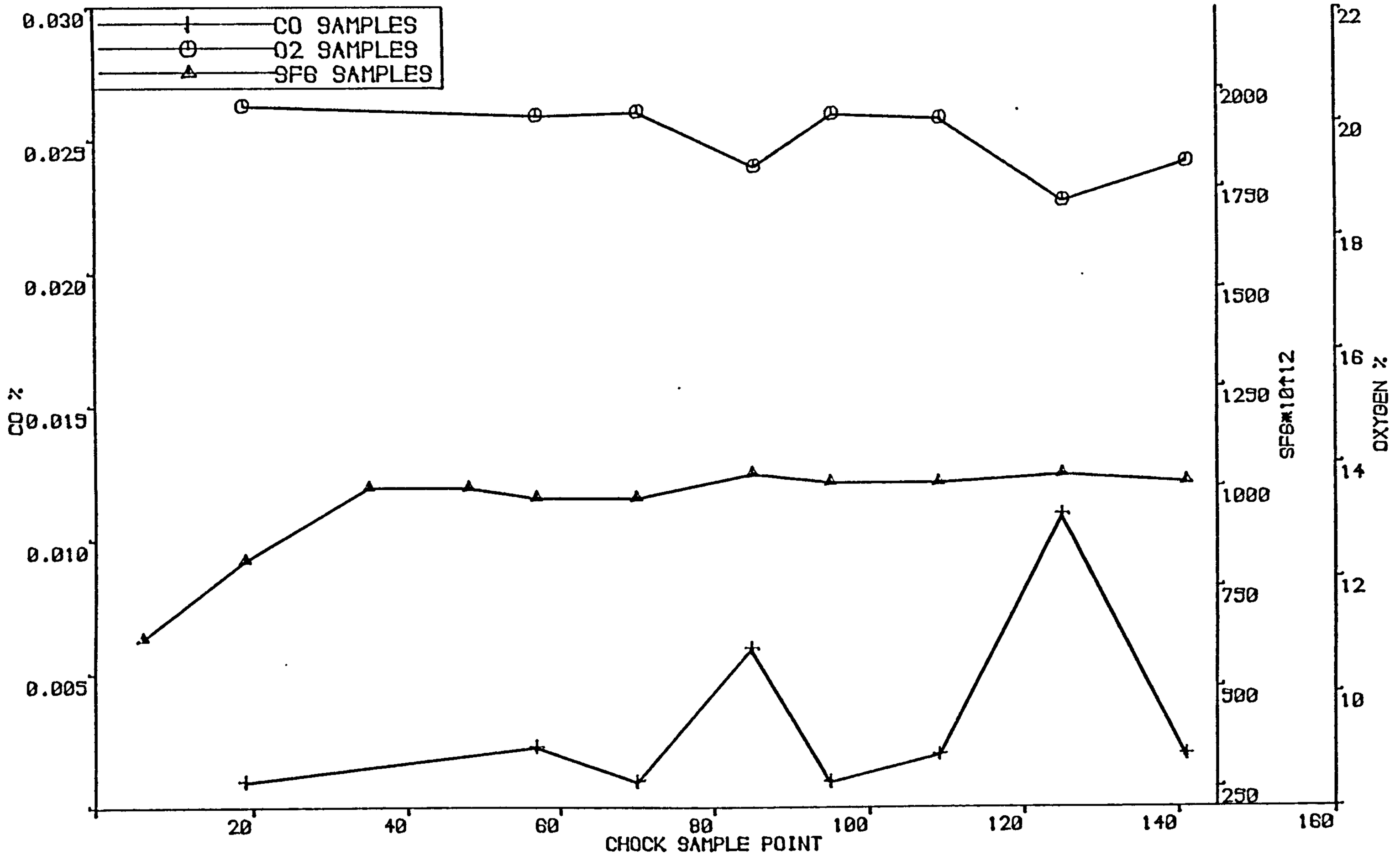


FIGURE 5.27 SAMPLES AT 17.10 ON 24/1

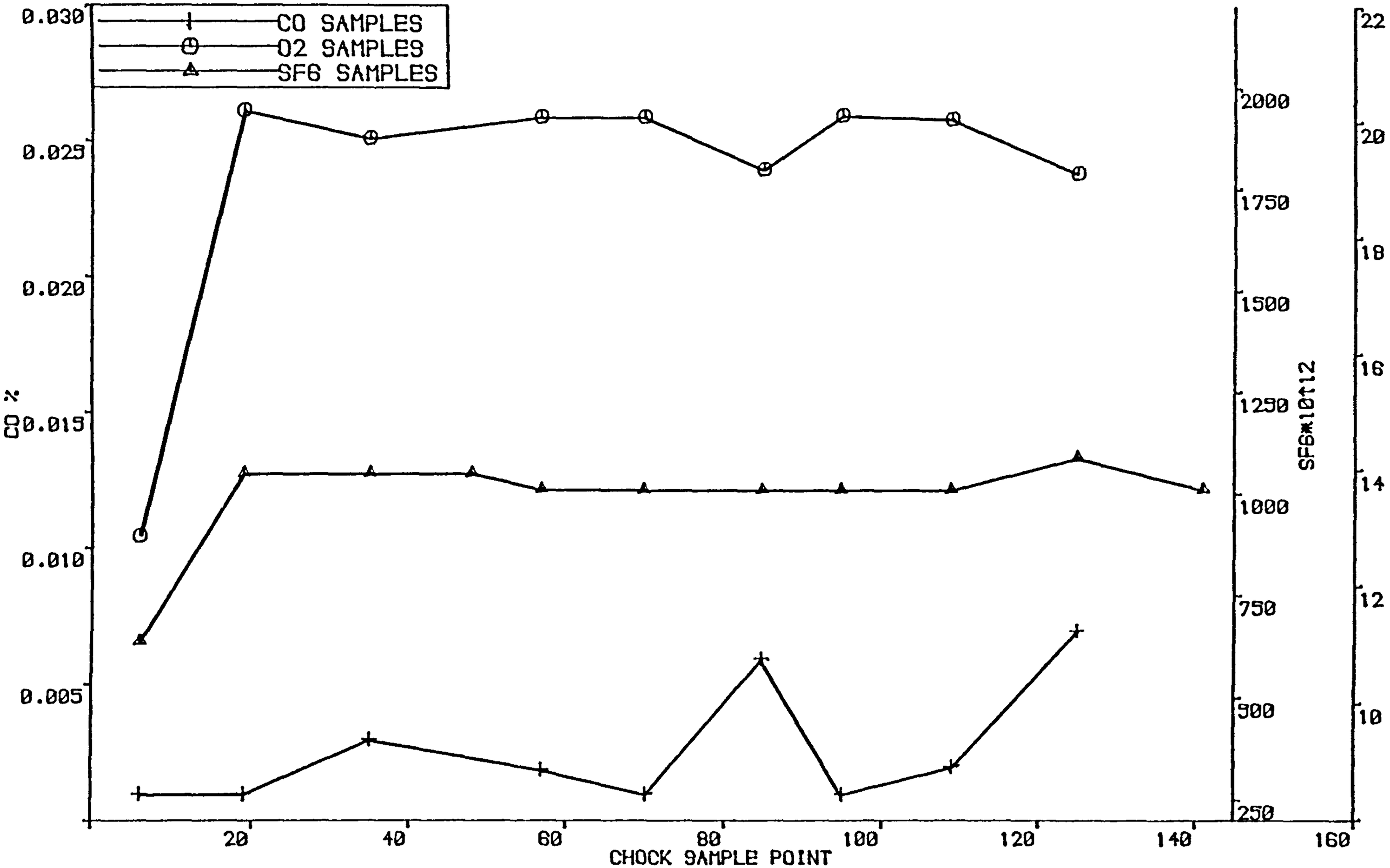


FIGURE 5.28 SAMPLES AT 19.50 ON 24/1

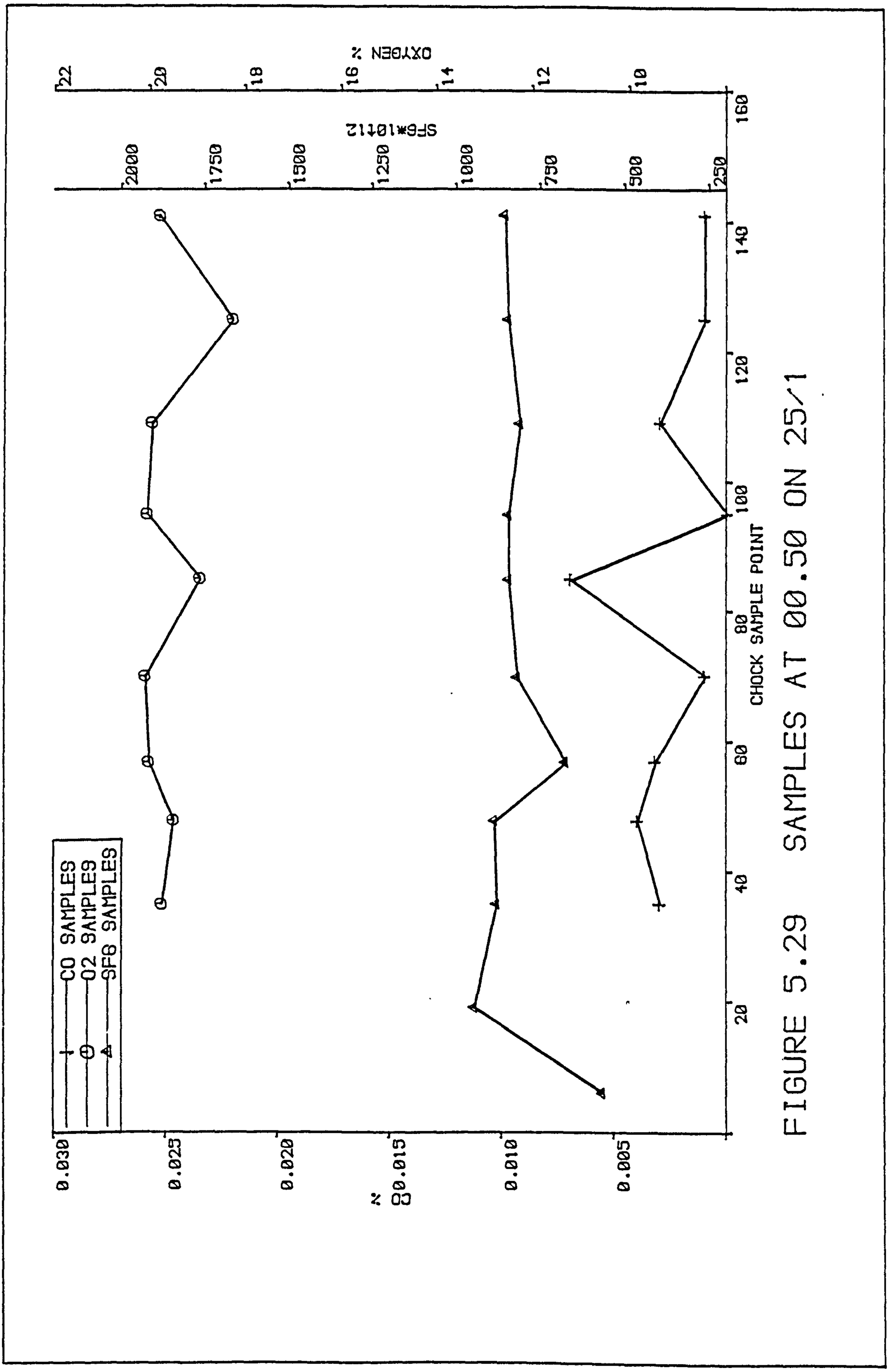


FIGURE 5.29 SAMPLES AT 00.50 ON 25/1

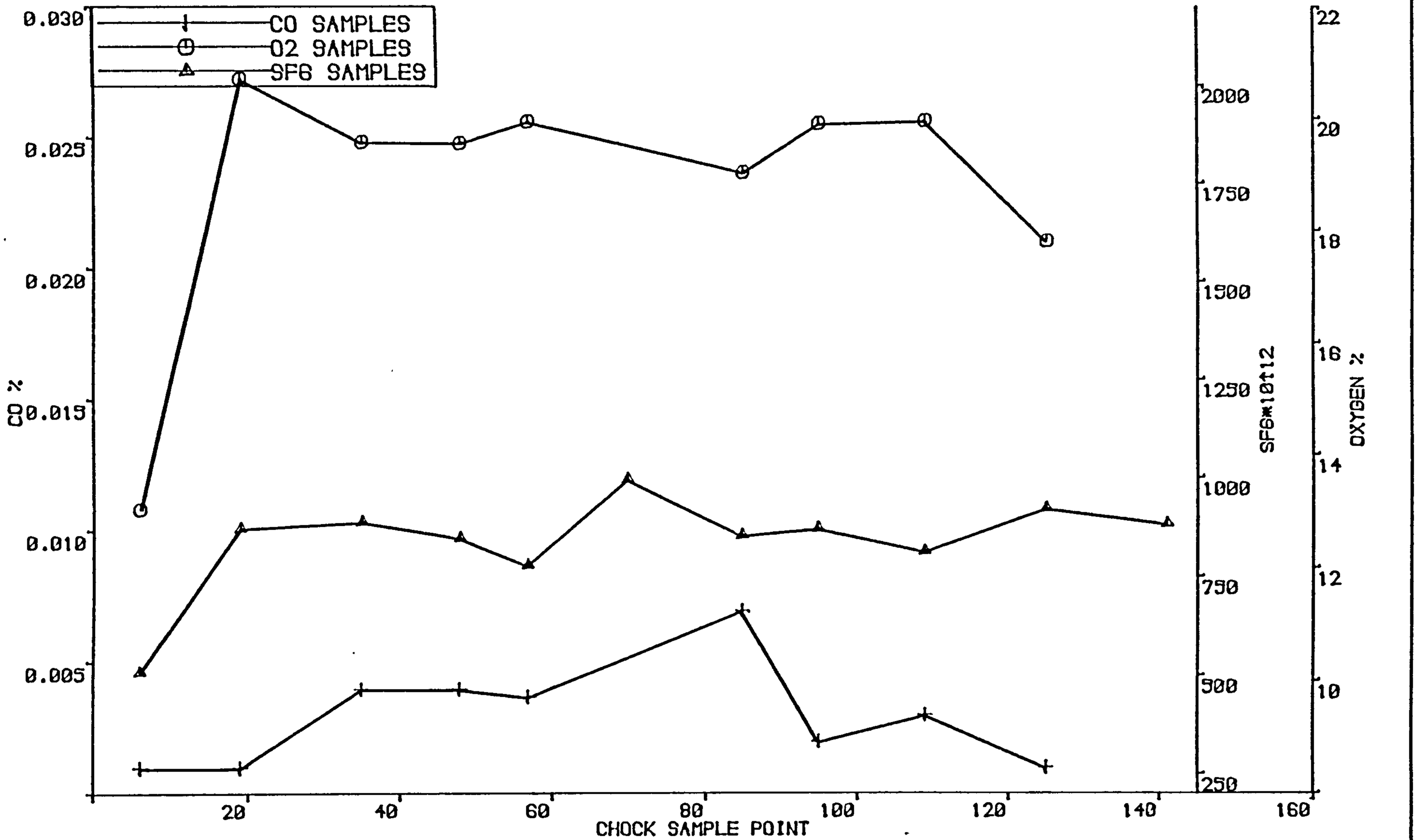


FIGURE 5.30 SAMPLES AT 03.50 ON 25/1

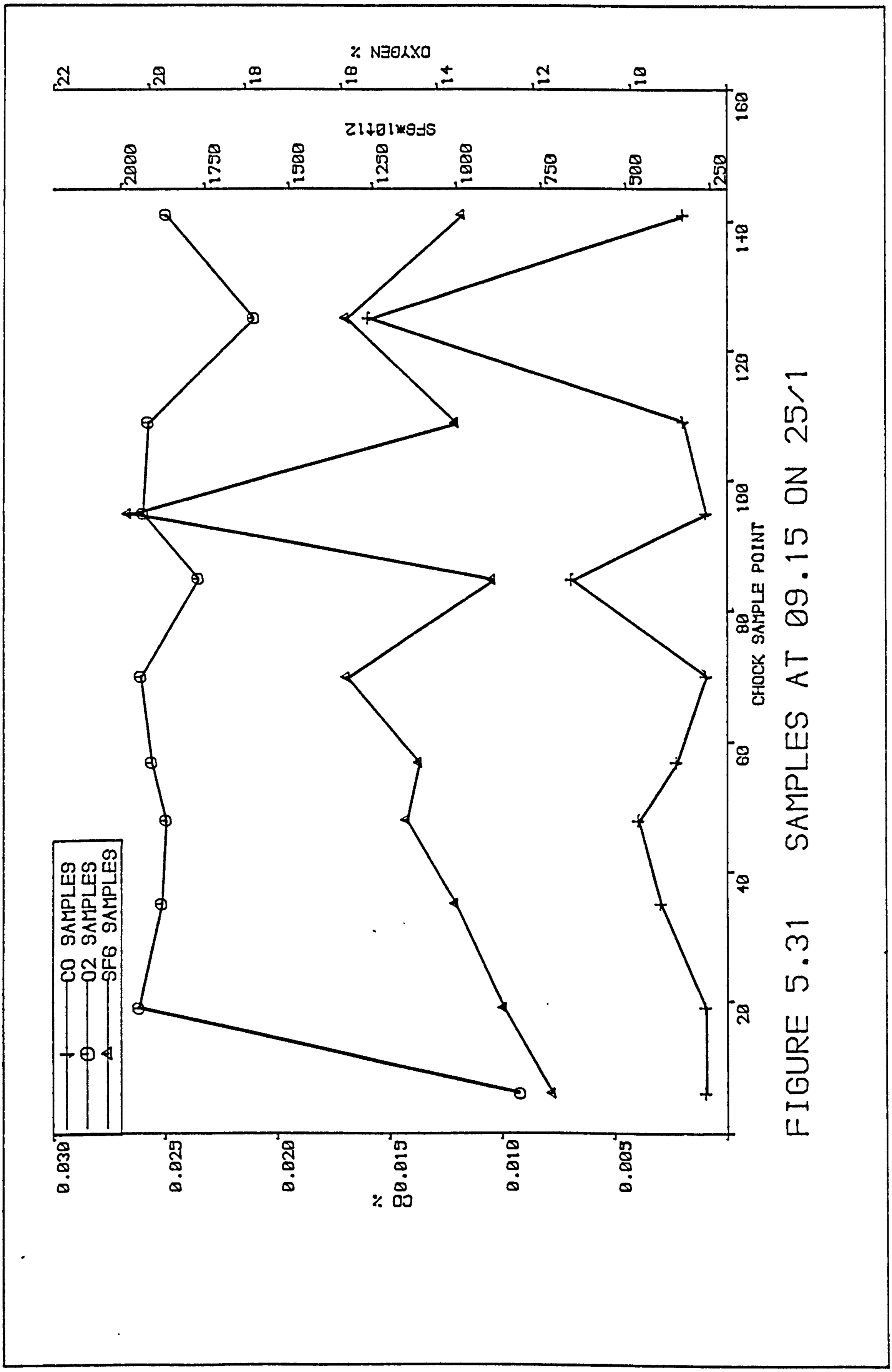


FIGURE 5.31 SAMPLES AT 09.15 ON 25/1

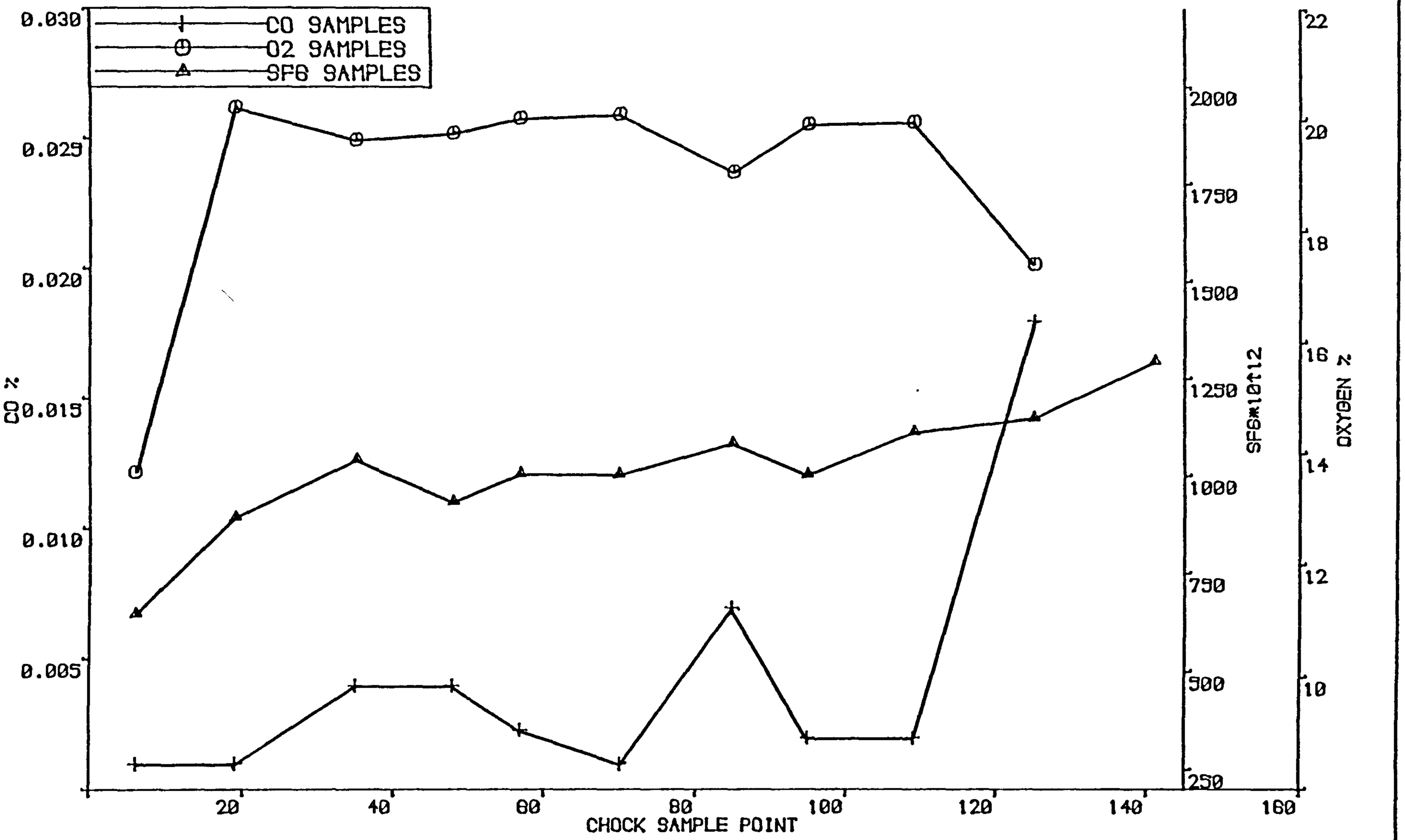


FIGURE 5.32 SAMPLES AT 11.45 ON 25/1

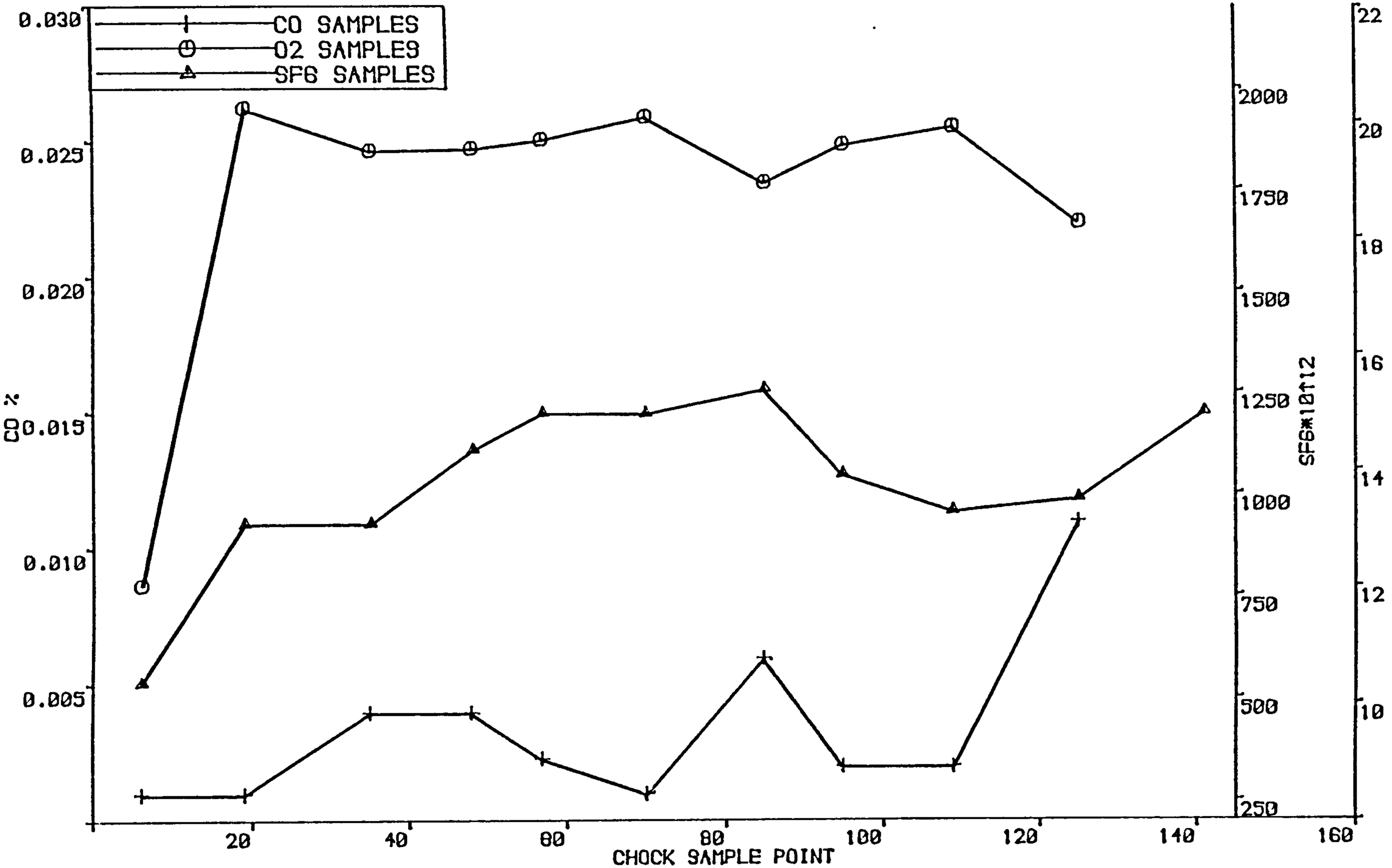


FIGURE 5.33 SAMPLES AT 17.30 ON 25/1

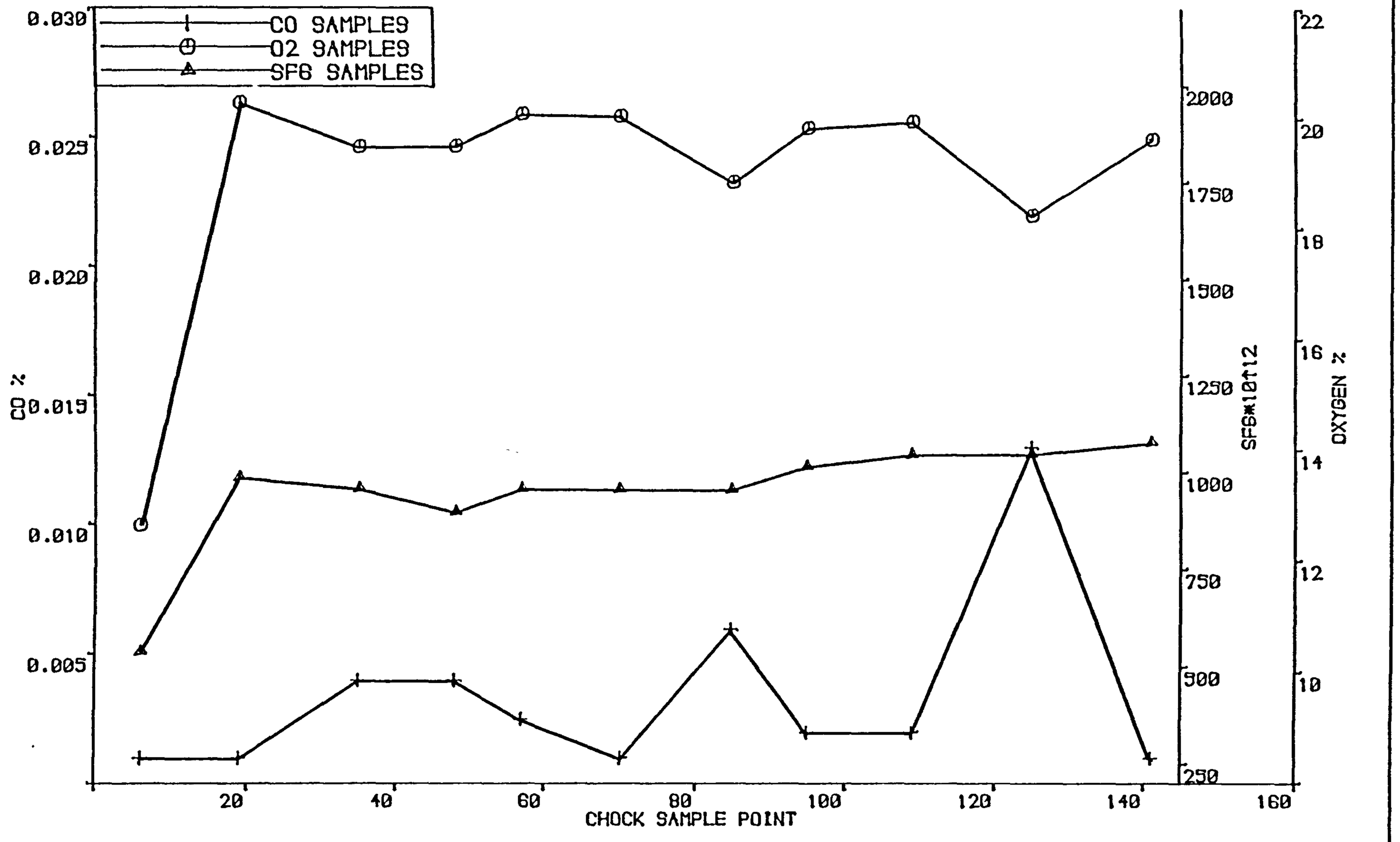


FIGURE 5.34 SAMPLES AT 20.30 ON 25/1

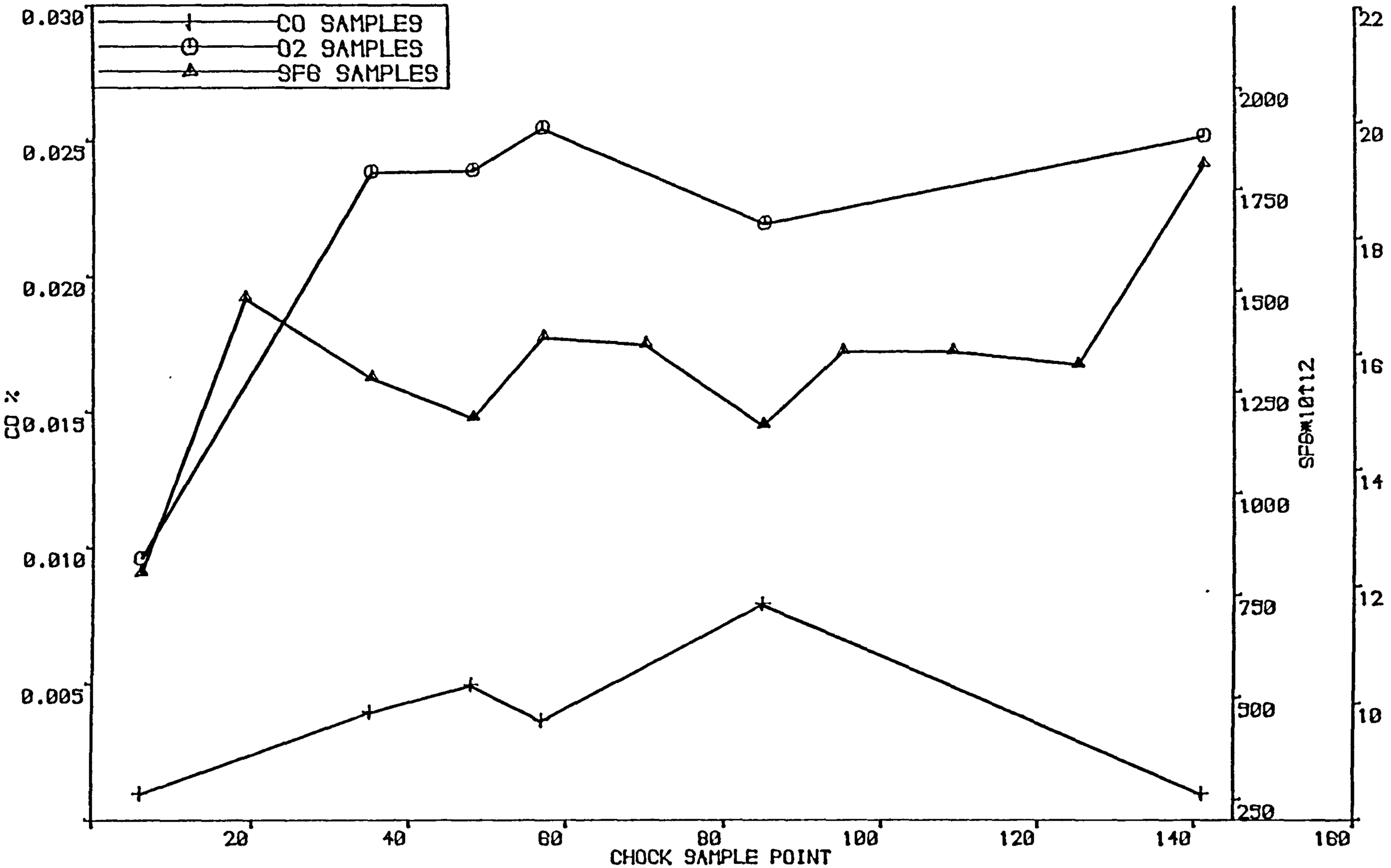


FIGURE 5.35 SAMPLES AT 00.30 ON 26/1

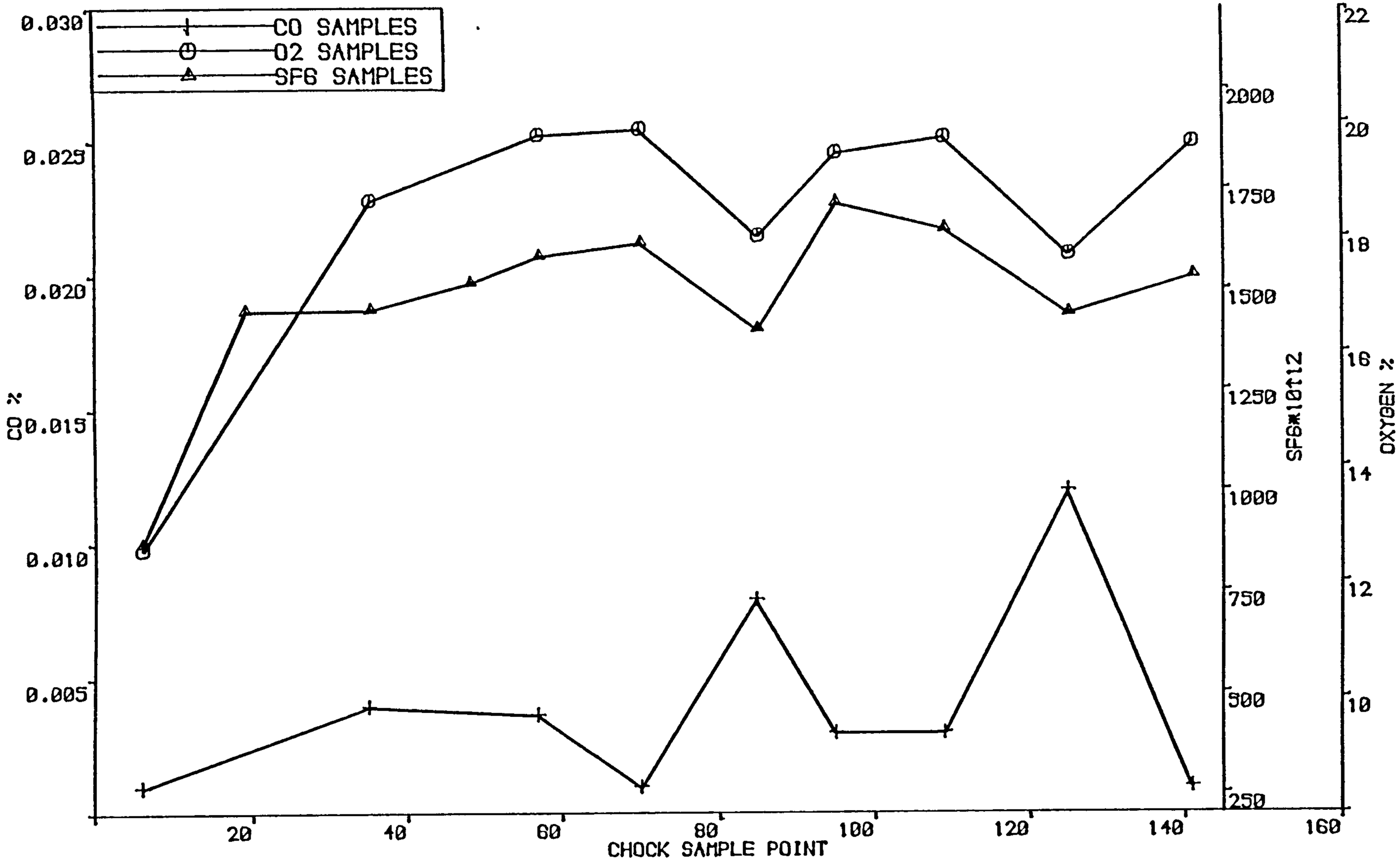


FIGURE 5.36 SAMPLES AT 04.00 ON 26/1

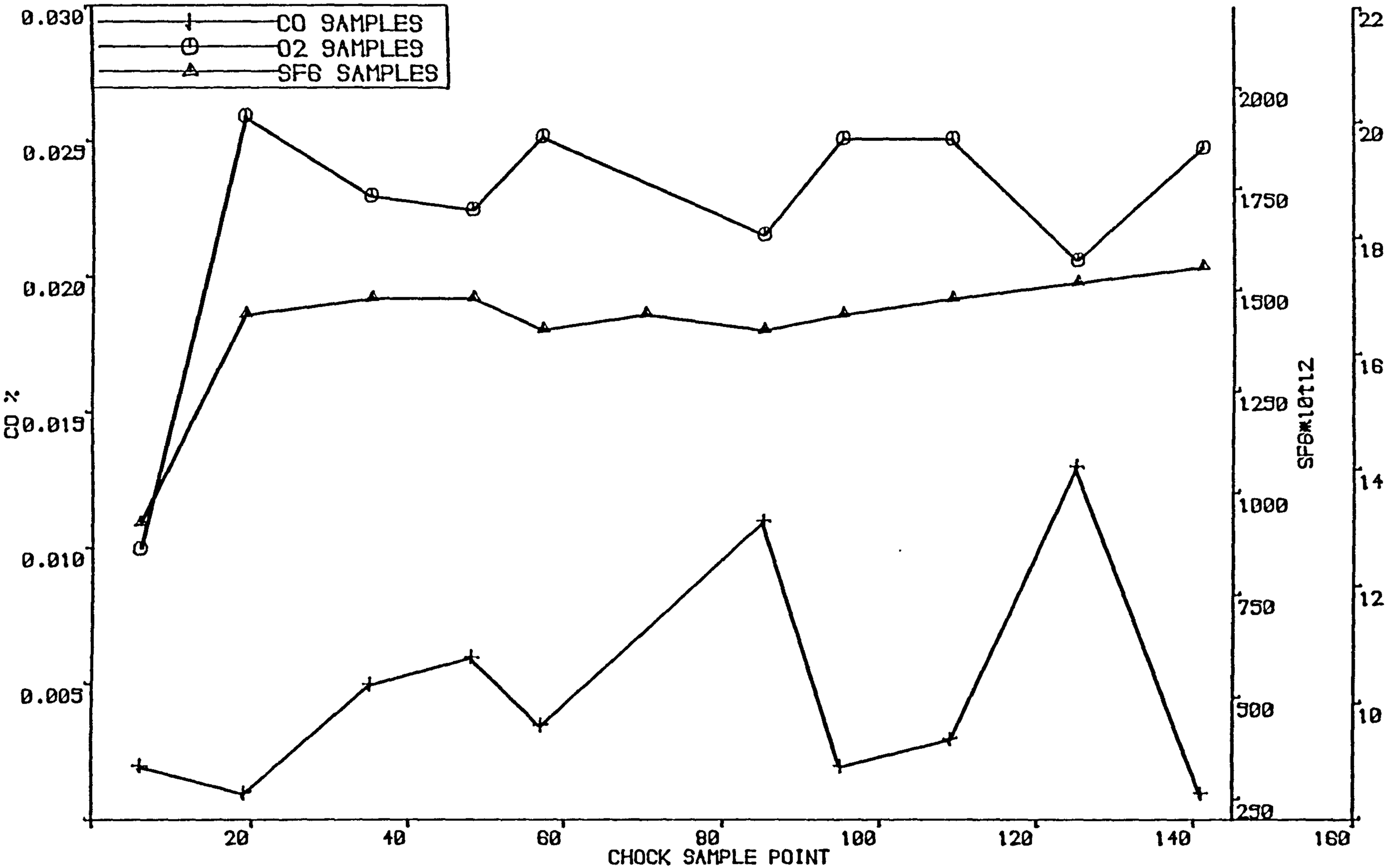


FIGURE 5.37 SAMPLES AT 08.55 ON 26/1

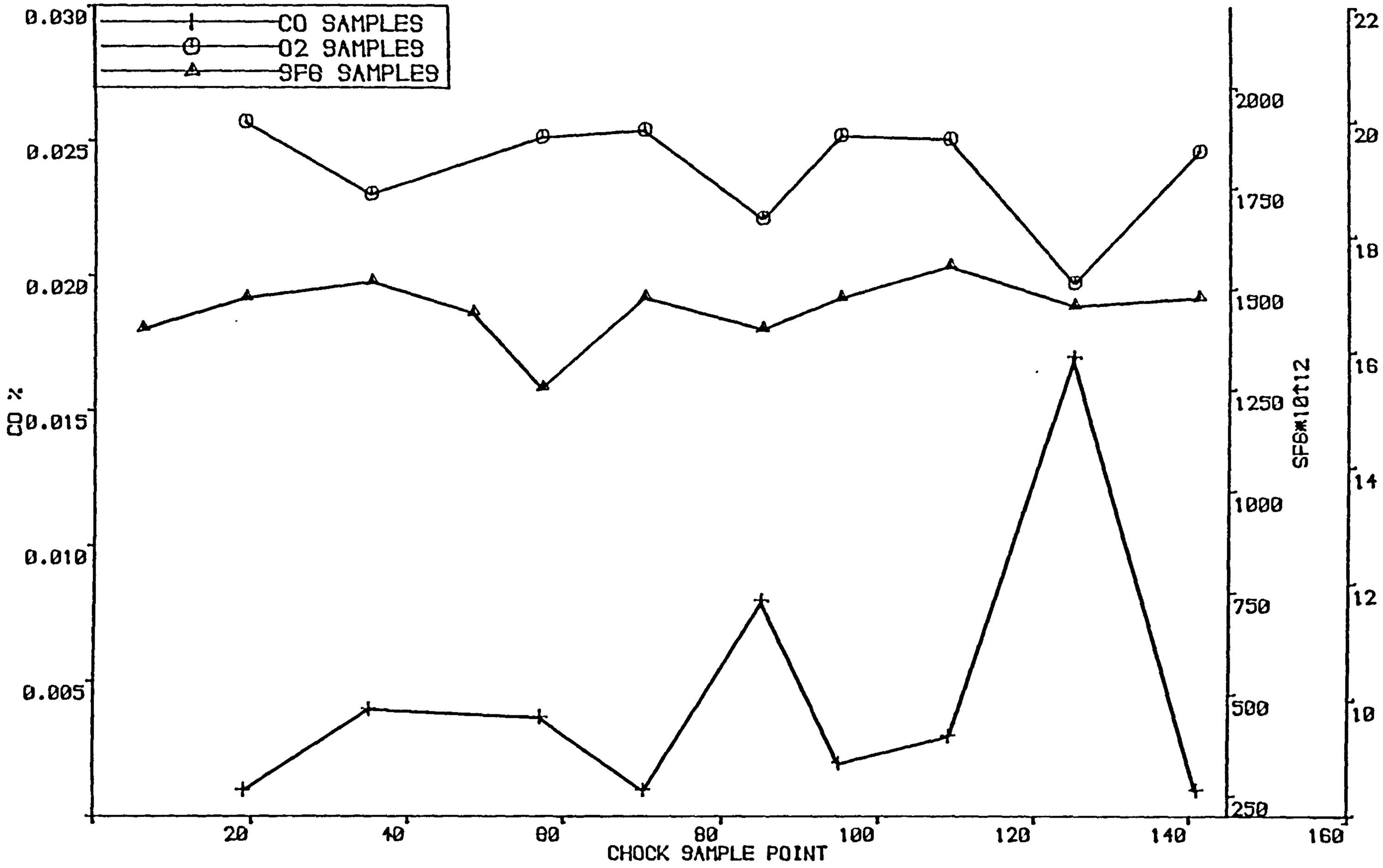


FIGURE 5.38 SAMPLES AT 11.35 ON 26/1

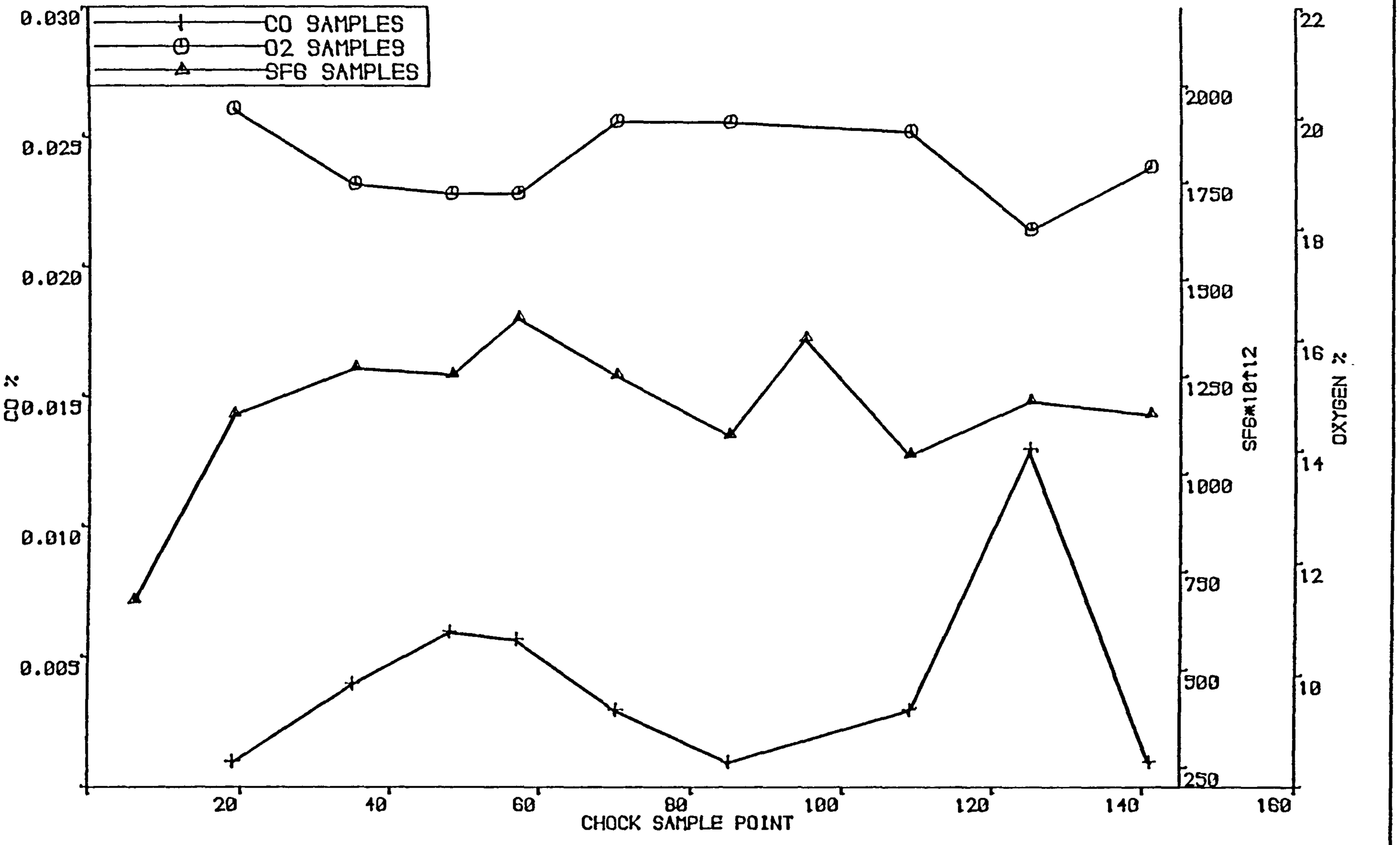


FIGURE 5.39 SAMPLES AT 16.45 ON 26/1

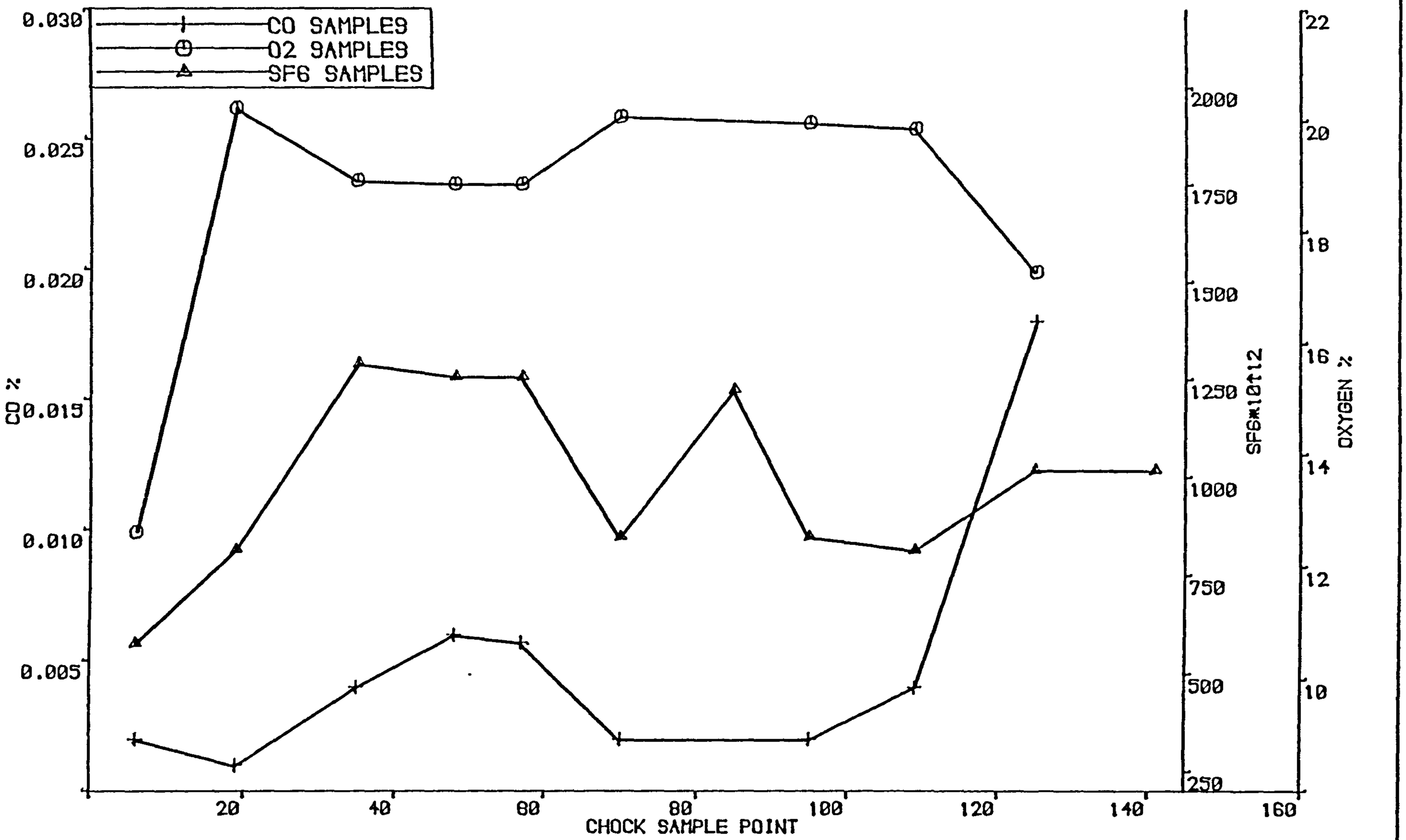


FIGURE 5.40 SAMPLES AT 20.25 ON 26/1

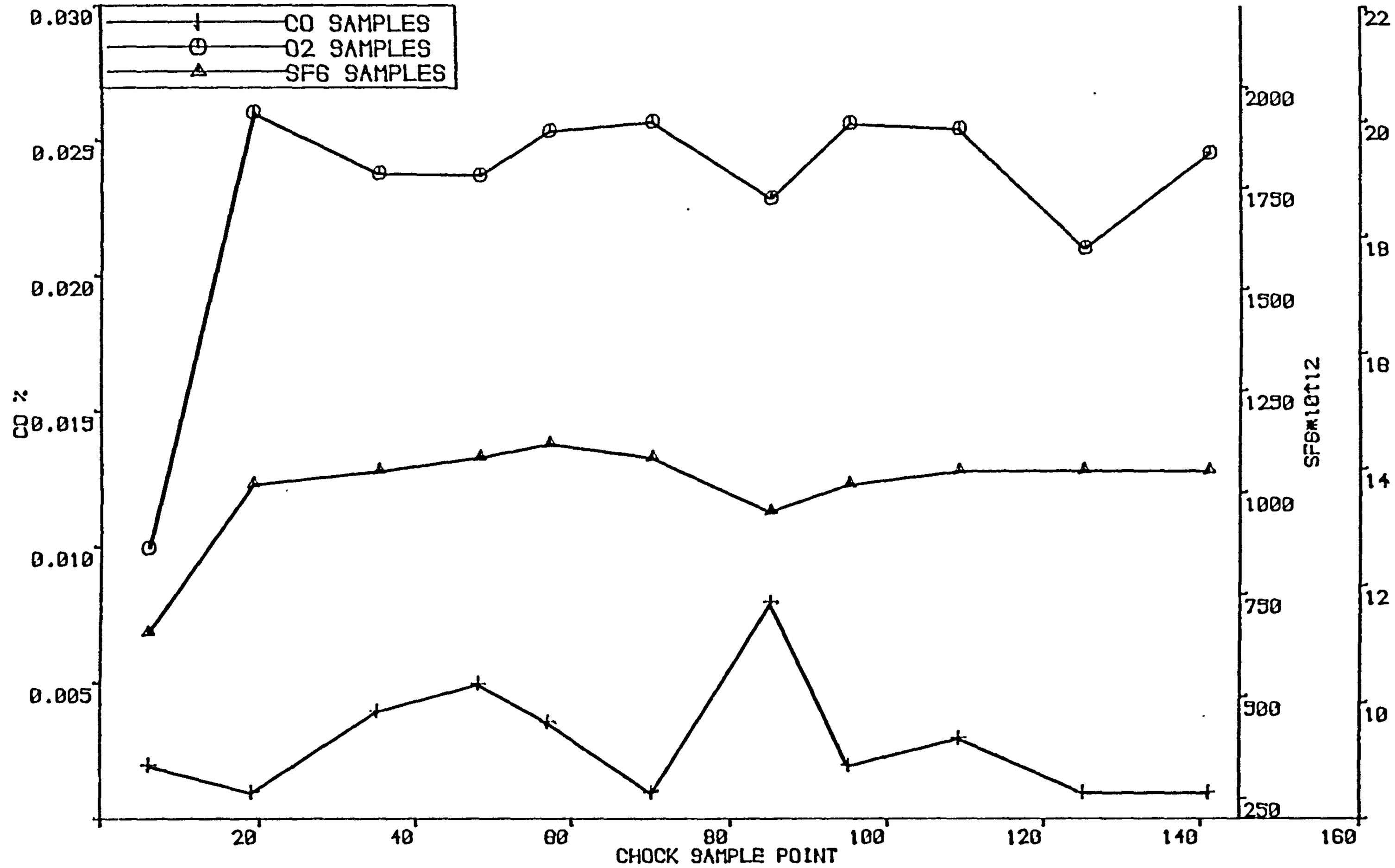


FIGURE 5.41 SAMPLES AT 00.45 ON 27/1

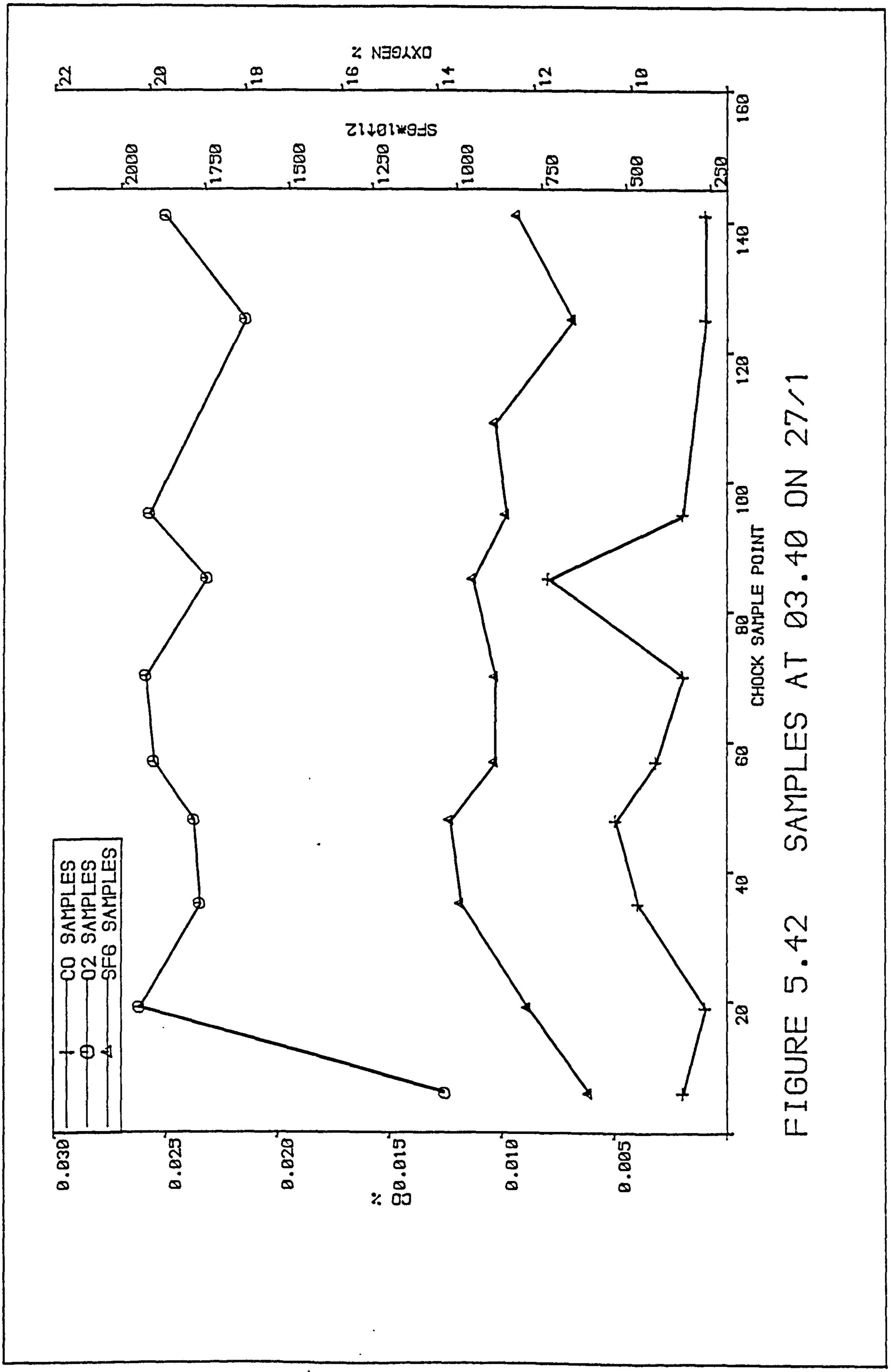


FIGURE 5.42 SAMPLES AT 03.40 ON 27/1
CHOCK SAMPLE POINT
% CO
CO SAMPLES
O2 SAMPLES
SF6 SAMPLES
OXYGEN %
SF6*10^12

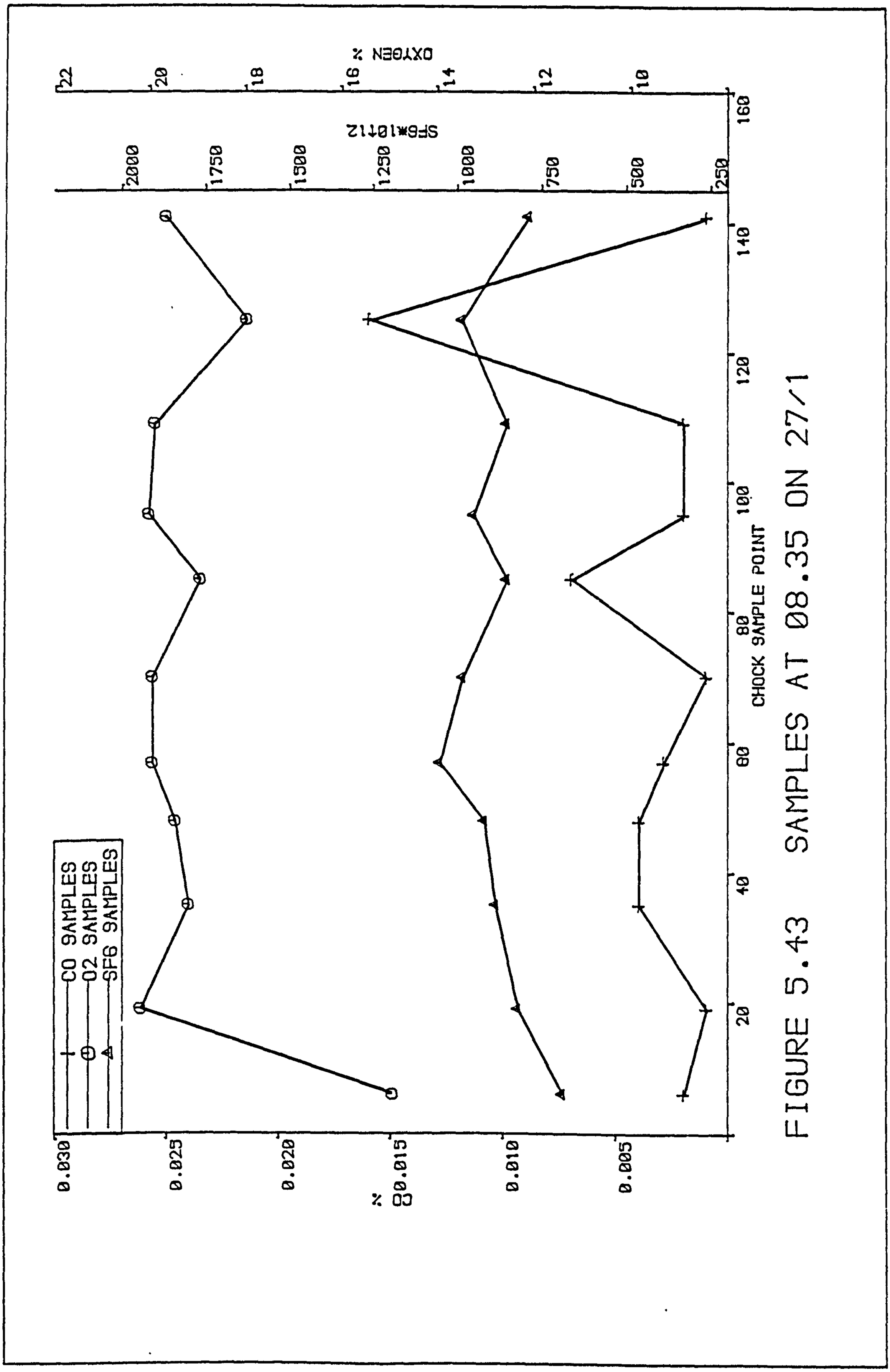


FIGURE 5.43 SAMPLES AT 08.35 ON 27/1

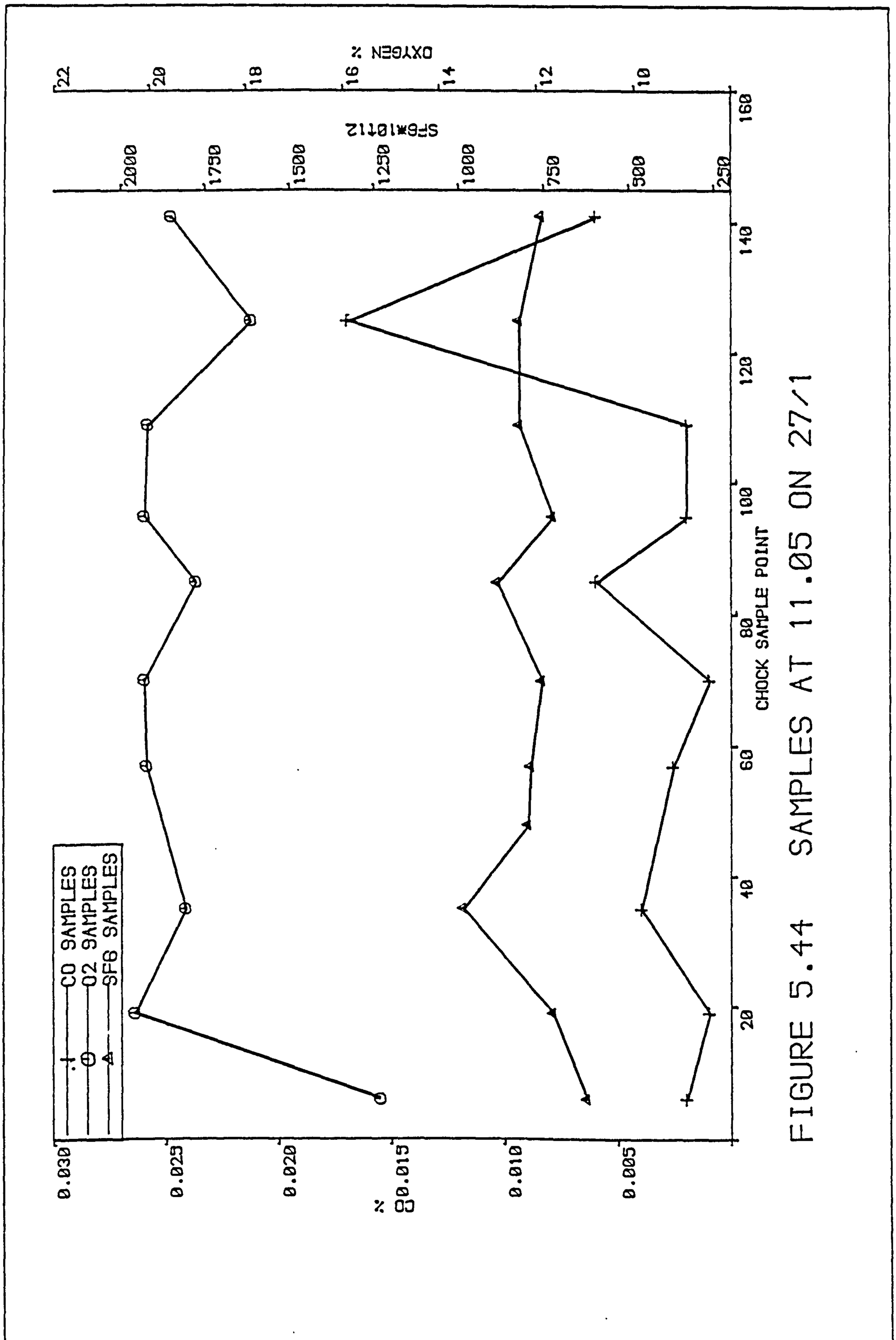


FIGURE 5.44 SAMPLES AT 11.05 ON 27/1

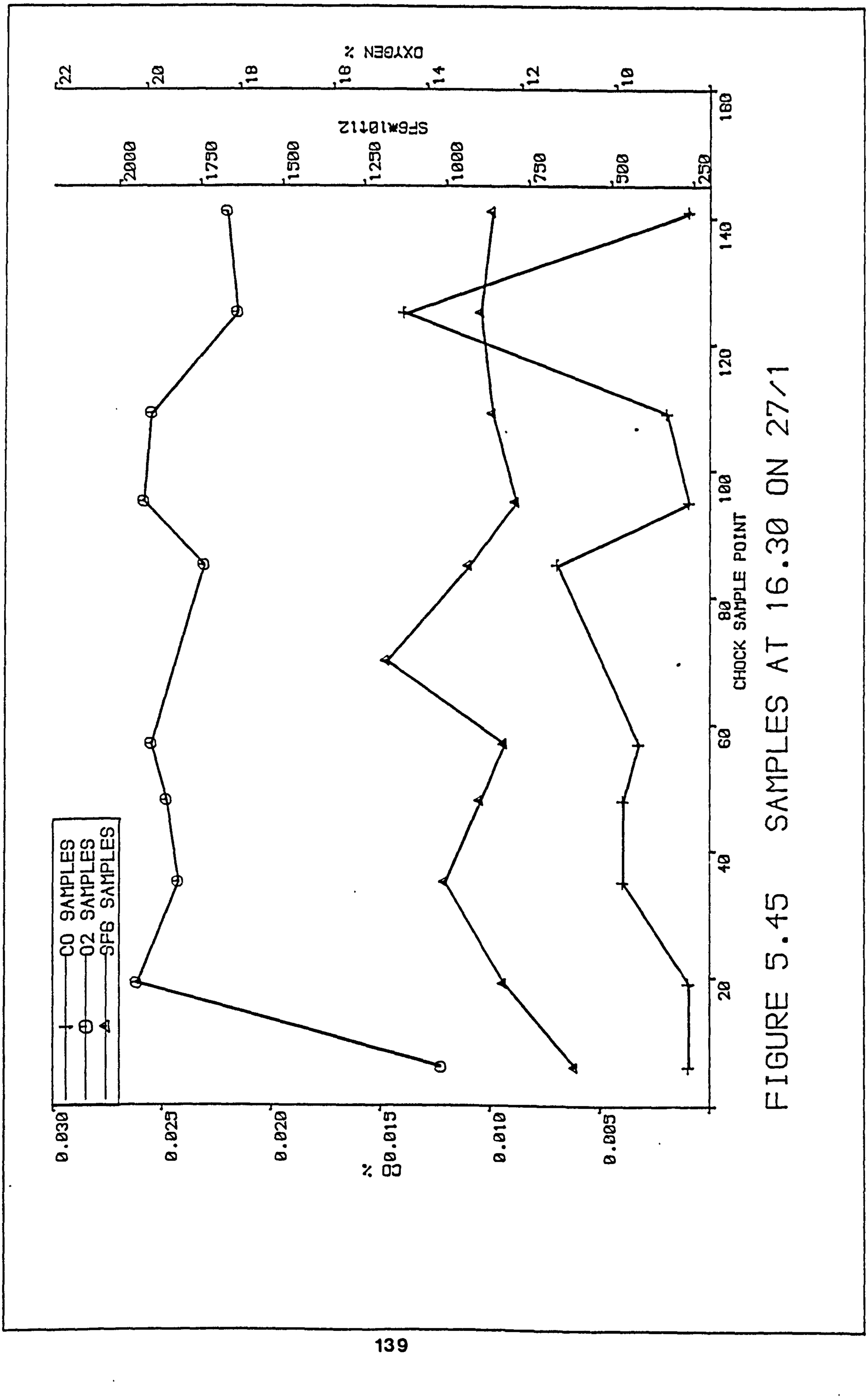


FIGURE 5.45 SAMPLES AT 16.30 ON 27/1

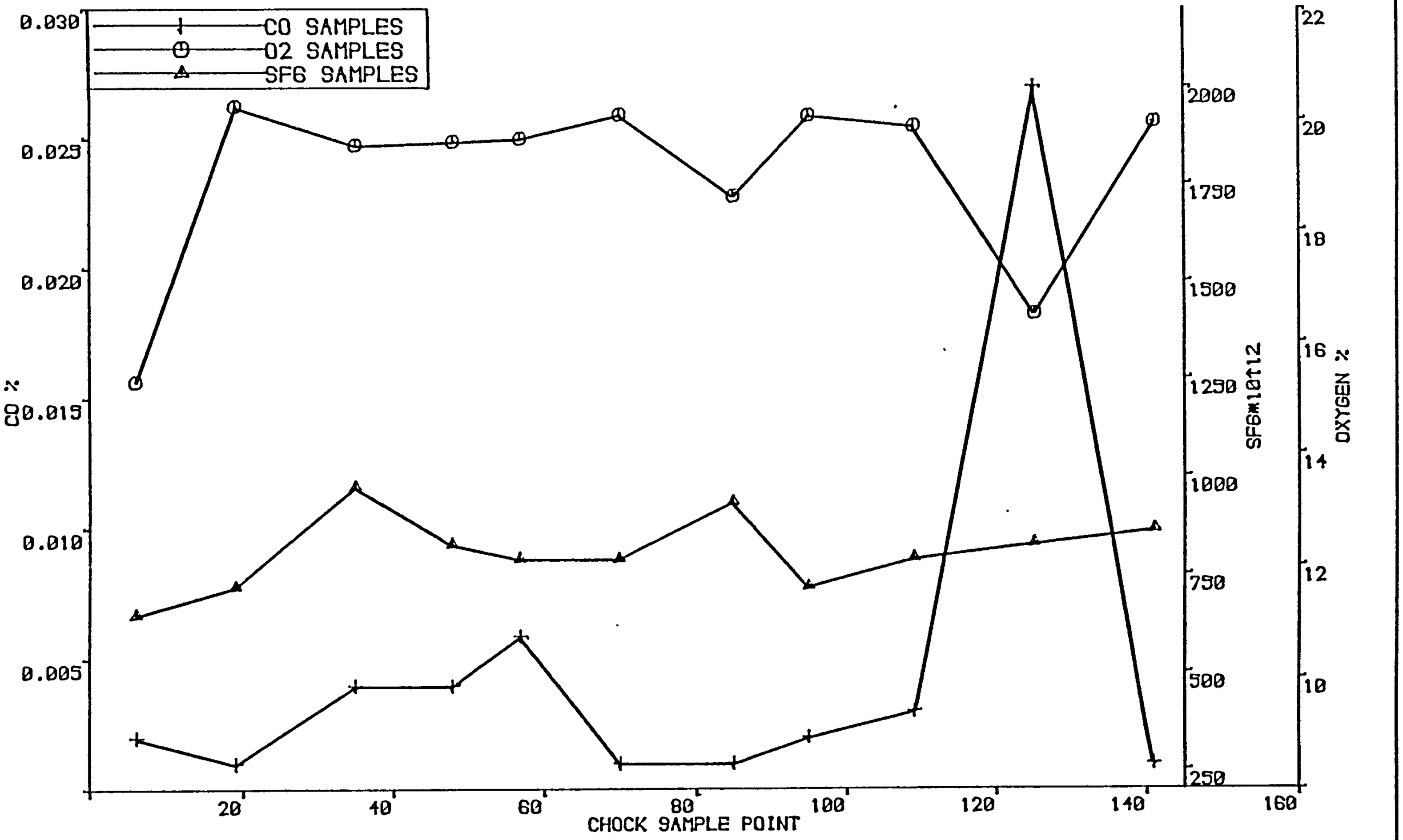


FIGURE 5.46 SAMPLES AT 01.30 ON 28/1

5.7 DISCUSSION OF RESULTS

Very shortly after the SF_6 began to be injected with the nitrogen, the general body air samples taken at the 10m ripping lip sample point in the intake showed a high level of SF_6. The level of SF_6 then rose and fell inversely with the barometer. The oxygen and carbon monoxide traces show some correlation with barometric changes. SF_6 was injected into the goaf at a rate equivalent to 3300 parts per 10^{12} in the general body. The sample analysis shows that between 25% and 60% of the SF_6 injected into the waste with the nitrogen appeared in the intake general body sample.

SF_6 appeared at high levels in the samples taken from the waste along the face line from the very first sample taken. This is almost certainly because of the zone of airflow behind the chocks, as described by Pickering(48). The effects of SF_6 in the general body will tend to dominate any SF_6 that appeared through goaf migration. The levels are lower than in the intake general body, but still followed the same, inverse relationship with the barometer. The oxygen levels in the face samples showed an almost exact rise and fall with the barometer. The oxygen level was lower than in the general body, while the carbon monoxide level was higher. The changes in gas concentrations exceed what would be expected purely from dilution by the injected nitrogen from the other nitrogen injection holes travelling in the airflow zone behind the face. Oxygen deficient air was moving through the goaf, and oxidation was occuring even at the fringes of the waste to produce

this effect. Although there is some scatter on the CO analysis, the trend was for the CO to rise and fall inversely with the barometer, as the SF_6 does.

There was an obvious reduction in oxygen and CO levels at the intake packside sample, largely due to the dilution by nitrogen. This indicates that much of the nitrogen injected over the pack, travelled down the waste side of the pack and into the general body face air, if it did not reemerge into the roadway from the strata before. It is probable that the nitrogen injected was only present in sufficient quanties to inhibit spontaneous combustion near to the packside.

At the return 10 metre sample point, almost exactly the same traces were obtained as at the intake. The oxygen level was slightly lower, while the CO level was higher. The intake general body level of SF_6 responded more quickly to barometric changes than the face samples. The sum of these produced a variable difference between the return and intake SF_6 levels.

The high levels of SF_6 around the cylinder and injection point suggested there was a leak in the SF_6 supply. To ensure that the SF_6 was not leaking from the injection system the bottle and reducer were replaced. Gas samples were taken around the the injection point to test for leaks, Table 15.The SF_6 level was found to be rising from a short distance outbye of the injection point. It is unlikely that this would result from a leak. The oxygen level fell as the SF_6 level rose, suggesting that nitrogen injected into the waste leaked out into the roadway very

close to the injection point through breaks in the strata around the roadway. Samples taken further inbye showed a fall in oxygen level as each of the nitrogen injection points were passed. The data shows that the SF_6 bottle and piping was not leaking. Much of the injected nitrogen is leaking back into the roadway from the injection point in the waste, probably through strata fractures around the roadway.

5.7.1 FACE PROFILES

Each graph shows a consistent shape, although this was subject to some distortion as the barometer changed. The oxygen and SF6 traces are very similar, while the carbon monoxide trace is almost a mirror image of these two.

The CO profile showed a steady rise then fall between the 19 and 70 chock sample points, and then peaked at the 85 and 125 sample points. The oxygen profile, after rising from the low percentage at the intake packside due to dilution, was almost constant except for two minima at the 85 and 125 chocks. There was a slight depression on the intake side of the face, between 19 and 70 chocks, which corresponds with the rise in CO levels. In the early stages of the experiment the oxygen trace had a further minimum at 48 chock, along with a slight peak in the CO level. It was during this part of the experiment that the barometer fell to 970 mB. As the barometer rose, this minimum disappeared and did not return, although the level of CO still showed a peak at this point.

The SF_6 profile, as mentioned earlier, was affected by barometric changes. When the barometer was rising the profile was almost flat. When the barometer was falling the profile showed a rising tendency between the 19 and 141 chock sample points. In the early stages of the experiment the profile was rather erratic. This may be due to the exceptionally low barometer at the time, or the SF_6 migration may not have stabilised. As nitrogen was being injected for some time before the introduction of SF_6, the migration paths in the waste are likely to be well established. Even at this early stage of the experiment there were minima at 47, 85 and 125 chock sample points. These minima were not consistently repeated. They did not occur when the barometer was rising, but only when falling or in a depression, as happened between the morning of January 25th and night of January 26th. The rise in SF_6 level when the barometer fell indicates that migration was taking place through the waste to the face line. The quantity reaching the face was not large, there is a 10% increase at most in SF_6 level between the 19 chock sample point and that at 141 chock.

There are two possible explanations for the peaks of carbon monoxide on the face profile coinciding with depressions in the level of oxygen and SF6. Obviously some oxygen was consumed to produce the CO, but the fall in oxygen level was too large for this to be the sole cause. The level of depression of oxygen and SF6, especially at the 85 and 125 chock sample points, suggests dilution by nitrogen. If this was the case, then the CO appearing at these points could be from a single source, nearer to the intake. The CO

would be flushed along leakage paths to emerge at these sample points. The higher levels of CO between 19 and 57 chocks, and corresponding depression in oxygen levels, could suggest that oxidation was occurring in that region, but the products of combustion were removed to further along the face line. The slight depression in SF_6 level at around 47 chock indicates that a leakage path existed there.

The most significant aspect of the waste gas levels along the face line is the dilution of all the gases at the intake packside. This must be because the majority of the nitrogen that was injected over the packs is travelling down this corridor. It reemerges into the general body at the packhole. Very little travels through the waste. It is likely that spontaneous combustion was inhibited only in the area around the pack.

The following conclusions can be drawn:

1. The goaf, as a whole, behaves like a giant sponge under the influence of air pressure changes. As the pressure rises, air, and nitrogen if it is being injected is forced in. When the pressure falls, the gases in the waste are flushed out. Other short term pressure changes around a district, such as.opening and closing air doors, will have a similar pulsing effect. This would account for the occasional high or low gas concentration that goes against the general trend.

2. The levels of oxygen concentration 1.5 metres into the waste confirms previous work which showed that air travels behind the chocks. Further work needs to be carried out to examine deeper into the waste to determine airflow and oxidisation zones.

3. To what extent the SF_6 on the face samples is due to air flowing behind the chocks is unknown. There is some leakage through the waste from the 1040 metre injection hole, as the level of SF_6 at 141 chock is marginally higher than that at 19 in most cases.

4. The roadway sides should be sealed around nitrogen injection points to prevent nitrogen leaking back into the roadway.

5. There are leakage paths through the waste, which allow nitrogen to flush through to the face line.

6. The deeper investigation of the atmosphere in the waste would lead to a greater understanding of the path nitrogen takes through the waste once injected.

7. The amount of injected nitrogen that actually enters the waste from the hole investigated is dependent on the barometric pressure.

8. Most of the injected nitrogen travels down the corridor next to the pack and emerges onto the face.

9. Should a consistent method of true waste sampling be developed then spontaneous heatings may be detected earlier.

After salvage was completed, nitrogen injection was stopped. Seven weeks later a sudden and rapid rise of CO concentration was detected, and the district had to be sealed off. High levels of CO were detected about two thirds of the way down the face line.

5.8 CONCLUSION

This chapter has shown the behaviour of nitrogen injected into the waste. The amount of nitrogen entering the waste is highly dependent on the changing barometric pressure.

There is flow down the unconsolidated corridor next to the pack, and immediately behind the face line. The depth of the flow behind the face line could not be identified. The next chapter will describe an attempt to obtain waste gas samples at increasing distances into the waste, to investigate the composition of the waste gas and the depth of flow in the waste.

CHAPTER 6

UNDERGROUND MEASUREMENT OF THE AIR ZONE BEHIND A LONGWALL COAL FACE

6.1 INTRODUCTION

It is accepted that deep seated waste heatings need sufficient air moving through the waste to initiate and then sustain a spontaneous heating. The conditions to initiate a heating may be narrower than those required to sustain one once started.

There are believed to be several zones of airflow (Section 3.3.3). The transition of airflow from zone to zone will be gradual rather than incremental. The position of each zone in the waste is not known.

This chapter details work that was done to examine the flow patterns in the waste. Gas samples were taken from the waste using a tube laid on the floor as the face advanced. From the change in composition of the gas it was hoped that inferences could be drawn about the airflow pattern.

6.2 PREVIOUS WORK

The existence of airflow through the waste of a longwall face was recognised when the waste was strip packed (49). It was shown that air left the face between packs, but was forced to return to the face line by the next pack. Air bled through the gateside packs in the intake and return.

Fully caved wastes were introduced in the 1960's, and today practically all the wastes in the UK are caved. The airflow pattern through a fully caved waste is different to that in a waste which has been strip packed. The volume of air leakage was investigated by Pickering (48). The airflow was measured in the intake and return, and on the face line. The volume of flow through the waste was found by difference.

Measurements were taken on six faces, with a seam thickness ranging from 0.92m to 1.98m. The results showed that between 23% and 10% of the return air quantity passed through the waste. The proportion of leakage was inversely proportional to the seam thickness.

To verify Atkinson's equation for a fully mechanised coal face, the pressure drop and airflow were measured on a face 76cm thick. 43% of the measured district airflow passed through the waste.

The airflow was measured 60 yards outbye of the return ripping lip, 6 yards outbye of the rip and at seven points along the face from the rip. The pressure drop was measured between the ripping lip station and the first station on the face, and between the first and last airflow measurement points on the face.

The airflow in the waste behind the face was calculated from the difference between the face airflow and the mean air quantity in the return. The pressure difference in the waste was taken as the same as that along the face line.

Atkinson's equation for turbulent airflow in a mine roadway is $P=RQ^n$.

On the district examined, the face line value of n was 2.07, while in the waste n = 1.78. The majority of air leaking through the waste was found to reenter the return roadway within 60m of the return rip.

More recently it has become standard practice in areas of British Coal that are extracting seams that are liable to spontaneous combustion to install sample pipes over the pack in the return airway every 50m as the face advances. The carbon monoxide content of the waste gas reaches a constant level, and any rise from this is treated as indicating a heating. To counter this the intake roadway is sealed opposite to the sample pipe. Different faces have different characteristics in the time taken to reach a constant waste gas composition, and the levels of constituents. This shows that airflow in the waste differs, possibly due to different

geological conditions, packing methods and ventilating pressures. The results signify that the airflow in the waste is largely parallel to the face line (32).

In Canada a study of waste leakage was undertaken to improve the ventilation at the working face (50). Only 37% of the air supplied to the district flowed along the face line. This showed that leakage was almost parallel to the face, even at distances greater than 400m from the face.

Research into the composition of the waste gas was carried out in Russia (51) to reduce the risk of explosions in the waste. The faces investigated were all retreat workings between 1.4 and 1.7 metres thick. A rapid rise in methane content was observed. The pattern of concentration was affected by the permeability of the waste, and the ventilation layout. Especially notable was the scouring effect at the intake face end, and along the roadway sides in Z ventilation layouts.

6.3 WASTE GAS SAMPLING AT HUCKNALL COLLIERY

6.3.1 HUCKNALL COLLIERY

Hucknall Colliery, closed in 1986, was six miles north-west of Nottingham. The mine worked the Black Shale seam at a depth of 600 metres. This seam is regarded as being liable to spontaneous combustion. During the operating life of the mine there were several small heatings. The work was done on the K7's face.

This was 3800 metres from the pit bottom, and in May 1985 had advanced 800 metres from the face start line. The rate of advance was averaging 1.5m/day. The extracted seam height was 1.6 metres, with a small amount of roof coal left. The face length was 240 metres, and was equipped with six leg conventional powered supports. The district had encountered many small faults, with displacements of up to 2m. The area around the point where a fault intersected the roadway was sealed on the waste side with boarding and plaster. Monolithic packing was used to reduce waste leakage. Ventilation was homotropal. Waste sample pipes were laid over the packs every 50 metres in the intake and return. Methane drainage holes were drilled in the return, but the methane make was usually insufficient for the system to be used.

6.3.2 WASTE SAMPLING TECHNIQUE

Any method of sampling waste gas from the waste had to meet the requirements of being easily extended as the face advanced, and not causing any interruption of the normal coal winning operations. Due to cost limitations the equipment used had to be inexpensive, or material that would otherwise be discarded.

The initial method of sampling was to lay a PVC tube between two chocks and let this trail into the waste, extending it as the face advanced. The pipe was buried in fine coal to absorb the crush in the waste. Short pieces of copper tubing were used to join lengths of PVC together, these joins were airtight. To ensure that no fresh air was was introduced to the sample point in the

waste, the end of the pipe was sealed with a clamp.

Samples were taken daily, at the outbye end of the return, the intake and return ten metre sample points, a general body sample on the face at the waste tube and from the sample tube. Before sampling from the waste tube it was purged using the hand pump.

After the first set of results it became obvious that a method of protecting the sample tube from damage in the waste had to be used. A better method was developed in stages.

To provide an airtight, continuous sample tube, small bore (1/8 or 3/16 inch diameter) nylon pressure tubing was used. A join between two lengths was formed by using different sized tube, and superglue to secure the join. This formed a strong, airtight join.

The nylon tube was threaded down 3/8 or 1/2 inch internal diameter hydraulic pressure hosing. This sort of hosing is armoured, is as strong as steel piping, and so protected the sample tube in the waste. Lengths of hosing were joined together using conventional hydraulic joins pushed onto the ends, connected by a nipple.

Five to ten metre lengths of tubing and hosing were used. Initially this was kept on a spool attached to the side of a chock. The tubing was laid on the floor, and reeled off the spool as the chock was advanced. Later the excess tubing was coiled in between the back legs of the chock, and uncoiled as the chock advanced.

After a number of breaks in the tube just after a joint had entered the waste, the joins were made stronger. The hydraulic joins were clamped onto the ends of the hose, and the whole joint was sleeved by a short length of pipe, which was filled with silicone sealer. Despite this, it proved impossible to sample from a depth greater than 15 metres into the waste.

The trace from the pit top barometer was collected to examine the effect that the changes in barometric pressure had on the composition of the gas samples.

An air quantity survey was conducted to find the quantity of the air flowing through the waste at the sample point.

6.3.3 RESULTS

Samples were taken from the first PVC tube up to 730cm into the waste. Table 6.1 shows the gas analysis of the general body samples, and Table 6.2 the analysis of the waste tube samples. Figure 6.1 is a graph of the Graham Ratio and carbon monoxide level plotted against the distance into the waste of the sample point, and Figure 6.2 is a graph of the oxygen, methane and carbon dioxide levels.

The repeated attempts to develop a workable method of sampling led to results from a large number of points up to 15 metres into the waste to be obtained. These are presented in Table 6.3. The general body samples did not change significantly and are not presented.

The analyses for carbon monoxide, Graham Ratio, carbon dioxide, methane and oxygen are plotted on scattergrams, Figures 6.3 to 6.7. These have had linear and quadratic regression curves fitted and plotted by computer.

It was not possible to take samples from depths greater than 15 metres.

The general body samples around the district did not change significantly. The full analyses for the general body samples are in Appendix 1, as is the barometer trace for the sampling period.

The ventilation survey results follow the tables and plots of the gas analyses.

DATE	SAMPLE LOCATION	CO2	CH4	0	N	CO	GRAHAM RATIO
23 MAY	INTAKE	0.08	0.08	20.75	79.09	0.0003	0.16
F.ADV.	SAMPLE TUBE G.B.	0.09	0.14	20.73	79.04	0.0004	0.20
70 CM	10 YDS. RETURN	0.09	0.16	20.70	79.05	0.0005	0.22
	OUTBYE END RETURN	0.13	0.21	20.58	79.08	0.0008	0.22
24 MAY	INTAKE	0.15	0.28	20.62	78.95	0.0003	0.10
F.ADV.	SAMPLE TUBE G.B.	INSUFFICIENT SAMPLE					
120 CM	10 YDS. RETURN	0.12	0.42	20.63	78.83	0.0004	0.16
	OUTBYE END RETURN	0.18	0.45	20.48	78.89	0.0008	0.20
30 MAY	INTAKE	0.04	0.02	20.84	79.10	0.0003	N/A/
F.ADV.	SAMPLE TUBE G.B.	0.04	0.05	20.82	79.09	0.0003	0.25
220 CM	10 YDS. RETURN	0.05	0.07	20.77	79.11	0.0004	0.24
	OUTBYE END RETURN	0.09	0.15	20.68	79.08	0.0007	0.27
3 JUNE	INTAKE	0.11	0.15	20.73	79.01	0.0003	0.16
F.ADV.	SAMPLE TUBE G.B.	0.10	0.18	20.72	79.00	0.0003	0.15
220 CM	10 YDS. RETURN	0.10	0.19	20.71	79.00	0.0004	0.19
	OUTBYE END RETURN	0.13	0.31	20.56	79.00	0.0007	0.19
4 JUNE	INTAKE	0.09	0.13	20.74	79.04	0.0003	0.16
F.ADV.	SAMPLE TUBE G.B.	0.10	0.21	20.72	78.97	0.0003	0.16
270 CM	10 YDS. RETURN	0.11	0.25	20.69	78.95	0.0004	0.18
	OUTBYE END RETURN	0.15	0.35	20.54	78.96	0.0007	0.19
5 JUNE	INTAKE	0.07	0.08	20.80	79.05	0.0003	0.23
F.ADV.	SAMPLE TUBE G.B.	0.08	0.16	20.77	78.99	0.0003	0.20
410 CM	10 YDS. RETURN	0.09	0.19	20.74	78.98	0.0004	0.22
	OUTBYE END RETURN	0.12	0.29	20.63	78.96	0.0006	0.21
6 JUNE	INTAKE	0.07	0.08	20.74	79.11	0.0005	0.25
F.ADV.	SAMPLE TUBE G.B.	0.07	0.14	20.70	79.09	0.0006	0.25
550 CM	10 YDS. RETURN	0.10	0.28	20.64	78.98	0.0006	0.21
	OUTBYE END RETURN	0.12	0.32	20.54	79.02	0.0009	0.24
7 JUNE	INTAKE	0.06	0.05	20.77	79.12	0.0005	0.28
F.ADV.	SAMPLE TUBE G.B.	0.08	0.14	20.74	79.04	0.0004	0.21
730 CM	10 YDS. RETURN	0.09	0.20	20.66	79.04	0.0006	0.22
	OUTBYE END RETURN	0.12	0.32	20.57	78.99	0.0007	0.20
11 JUNE	INTAKE	0.10	0.16	20.68	79.06	0.0004	0.16
F.ADV.	SAMPLE TUBE G.B.	0.10	0.28	20.67	78.95	0.0005	0.21
180 CM	10 YDS. RETURN	0.10	0.28	20.64	78.98	0.0006	0.21
	OUTBYE END RETURN	0.13	0.34	20.52	79.01	0.0010	0.25

Table 6.1 First Phase General Body Gas Analyses

FACE ADVANCE (CM)	DATE	CO2 %	METHANE %	OXYGEN %	NITROGEN %	CO %	GRAHAM RATIO
0	23 MAY	0.09	0.14	20.73	79.04	0.0004	0.20
70	23 MAY	0.10	0.15	20.69	79.06	0.0005	0.21
180	11 JUNE	0.12	0.36	20.41	79.11	0.0012	0.23
220	24 MAY	0.16	0.41	20.49	78.94	0.0007	0.17
220	30 MAY	0.08	0.22	20.51	79.19	0.0015	0.33
220	3 JUNE	0.11	0.31	20.49	79.09	0.0014	0.31
270	4 JUNE	0.12	0.28	20.58	79.02	0.0008	0.24
410	5 JUNE	0.15	0.77	20.21	78.87	0.0021	0.31
550	6 JUNE	0.16	0.70	20.03	79.11	0.0029	0.32
730	7 JUNE	0.16	0.83	19.90	79.11	0.0035	0.34

Table 6.2 First Phase Gas Analyses

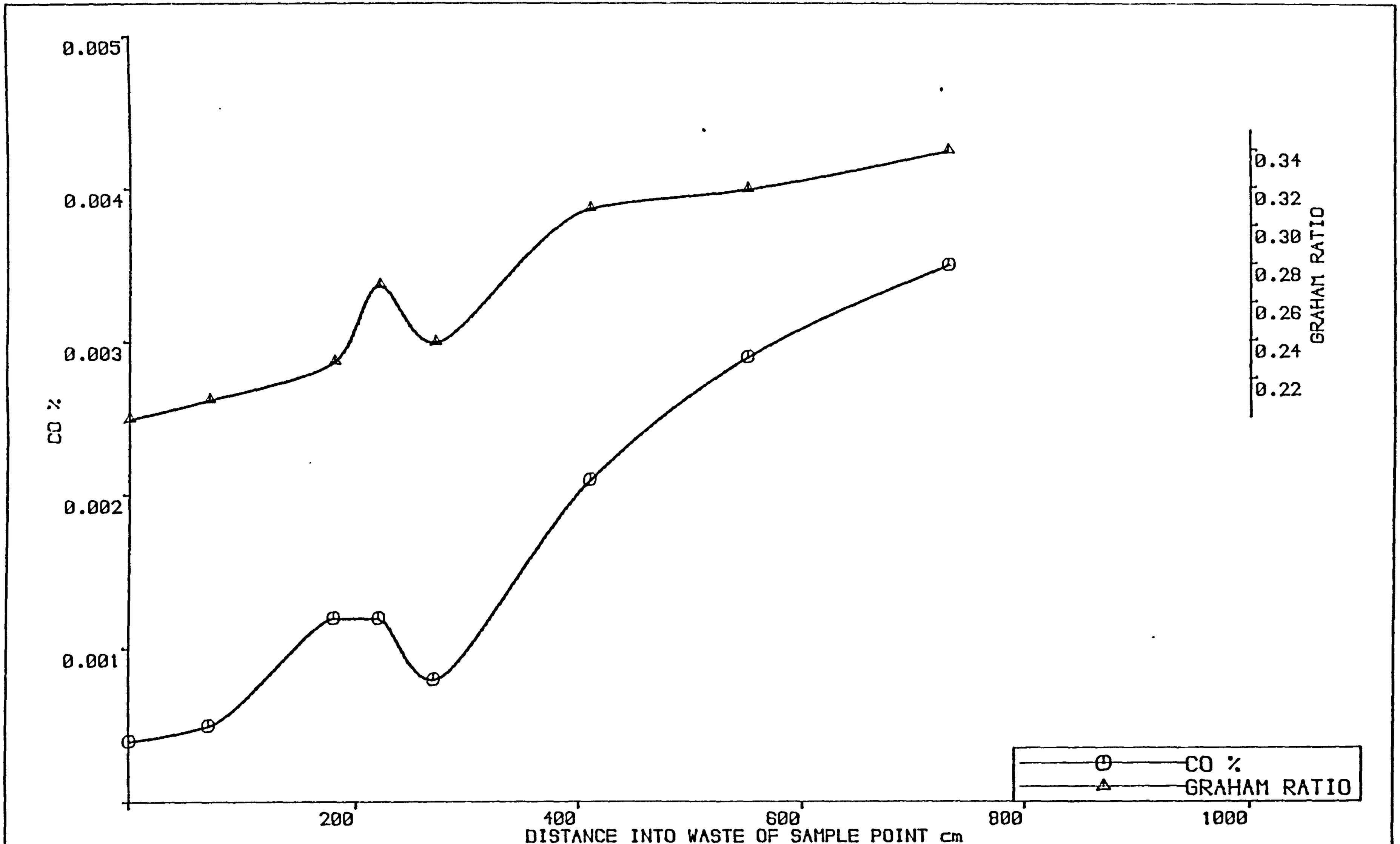


Figure 6.1 Graham Ratio & Carbon Monoxide Level vs. Distance into the Waste

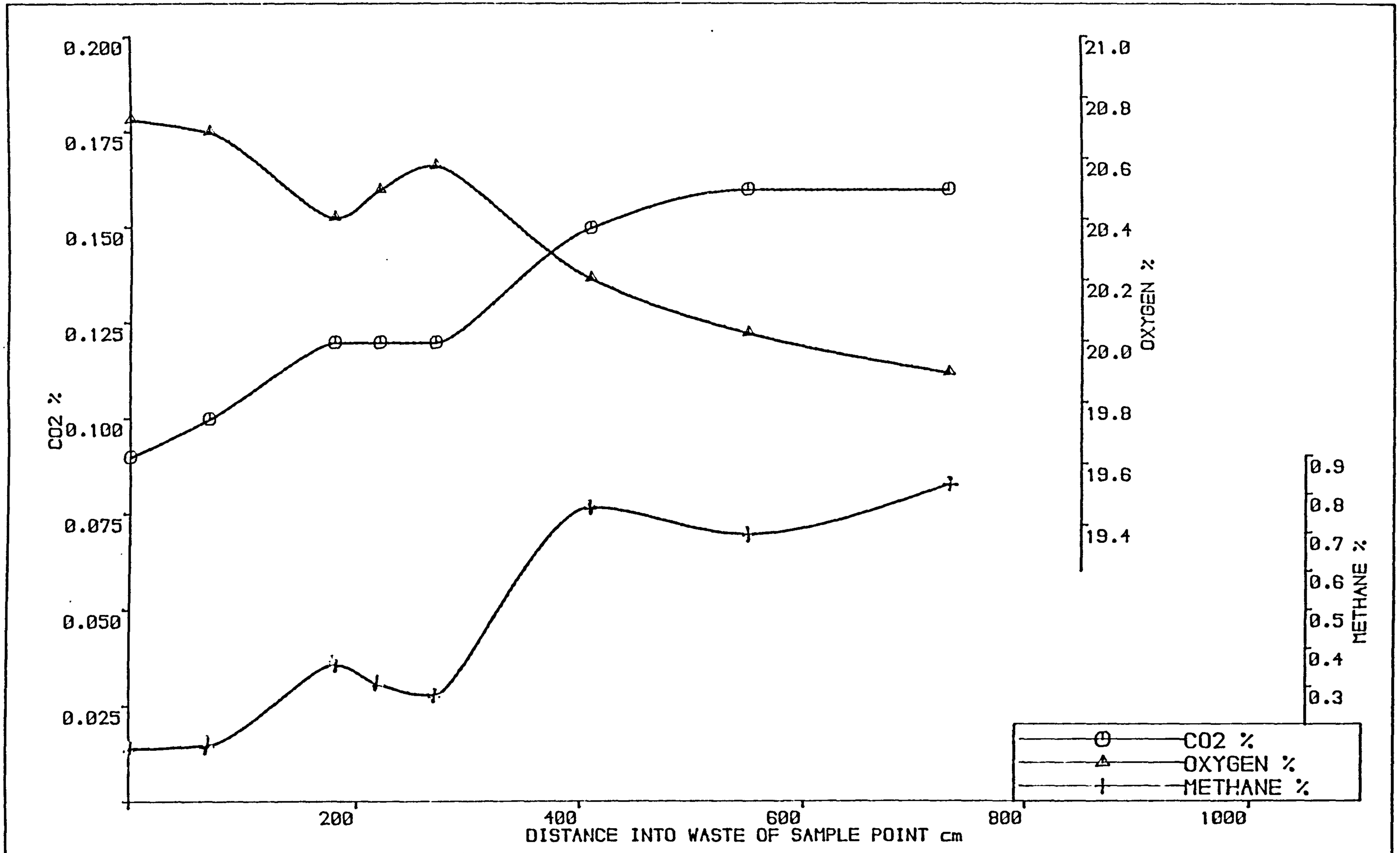


Figure 6.2 Oxygen, Methane, & Carbon Dioxide Levels vs. Distance into the Waste

FADV m	CO2 %	CH4 %	O2 %	N2 %	CO %	GRAHAM RATIO
0.50	0.08	0.10	20.76	79.06	0.0004	0.24
0.50	0.06	0.21	20.63	79.00	0.0008	0.28
0.50	0.12	0.22	20.62	79.04	0.0008	0.26
1.00	0.08	0.16	20.68	79.08	0.0007	0.27
1.00	0.09	0.26	20.66	78.99	0.0008	0.31
1.50	0.09	0.15	20.66	79.10	0.0007	0.25
1.70	0.16	0.47	20.37	79.00	0.0016	0.29
1.70	0.11	0.38	20.29	79.22	0.0020	0.29
1.80	0.12	0.36	20.41	79.11	0.0012	0.23
2.00	0.07	0.11	20.73	79.09	0.0006	0.29
2.00	0.20	0.42	20.27	79.11	0.0017	0.25
2.20	0.11	0.31	20.49	79.09	0.0014	0.31
2.20	0.08	0.22	20.51	79.19	0.0015	0.33
2.50	0.12	0.23	20.63	79.02	0.0007	0.24
2.50	0.20	0.13	20.56	79.11	0.0025	0.29
3.00	0.14	0.40	20.27	79.19	0.0024	0.34
3.00	0.10	0.20	20.52	79.18	0.0014	0.31
3.50	0.09	0.26	20.45	79.20	0.0013	0.25
3.50	0.16	0.38	20.52	78.94	0.0012	0.32
3.50	0.40	0.34	20.46	79.06	0.0014	0.30
3.50	0.10	0.22	20.52	79.16	0.0015	0.34
3.70	0.12	0.20	20.48	79.20	0.0016	0.33
4.00	0.11	0.29	20.52	79.08	0.0014	0.33
4.00	0.10	0.46	20.53	78.91	0.0010	0.28
4.00	0.11	0.55	20.43	78.91	0.0013	0.29
4.10	0.15	0.77	20.21	78.87	0.0021	0.31
4.20	0.17	0.54	20.12	78.17	0.0024	0.41
4.50	0.12	0.26	20.53	79.09	0.0012	0.30
4.50	0.10	0.32	20.40	79.18	0.0018	0.32
4.75	0.16	0.21	20.41	79.22	0.0018	0.32
5.00	0.22	0.56	20.30	78.92	0.0012	0.20
5.00	0.14	0.55	20.31	79.00	0.0018	0.31
5.10	0.13	0.31	20.35	79.21	0.0021	0.34
5.50	0.16	0.70	20.03	79.11	0.0029	0.32
5.75	0.14	0.40	20.40	79.06	0.0013	0.25
5.75	0.12	0.55	20.28	79.05	0.0020	0.30
6.20	0.15	0.41	20.27	79.17	0.0019	0.29
6.50	0.11	0.31	20.44	79.14	0.0014	0.27
6.50	0.15	0.43	20.23	79.18	0.0023	0.31
6.70	0.09	0.30	20.30	79.31	0.0024	0.30
7.00	0.10	0.38	20.24	79.28	0.0023	0.31
7.30	0.16	0.83	19.90	79.11	0.0035	0.34
7.50	0.09	1.13	19.90	78.87	0.0029	0.27
7.60	0.13	0.32	20.31	79.24	0.0022	0.33
8.10	0.13	0.27	20.43	79.17	0.0020	0.38
8.20	0.09	0.29	20.36	79.26	0.0019	0.30
9.00	0.14	0.42	20.09	79.35	0.0031	0.34
9.50	0.15	0.38	20.32	79.15	0.0023	0.36
9.60	0.17	1.17	19.88	78.77	0.0027	0.28
9.70	0.19	0.42	20.16	79.23	0.0033	0.40
10.50	0.15	0.77	20.02	79.05	0.0029	0.32
12.10	0.19	0.38	20.30	79.13	0.0027	0.42
13.00	0.17	0.95	19.76	79.12	0.0043	0.36
15.00	0.17	0.50	20.03	79.29	0.0033	0.38

Table 6.3 Gas Analyses From All Waste Samples

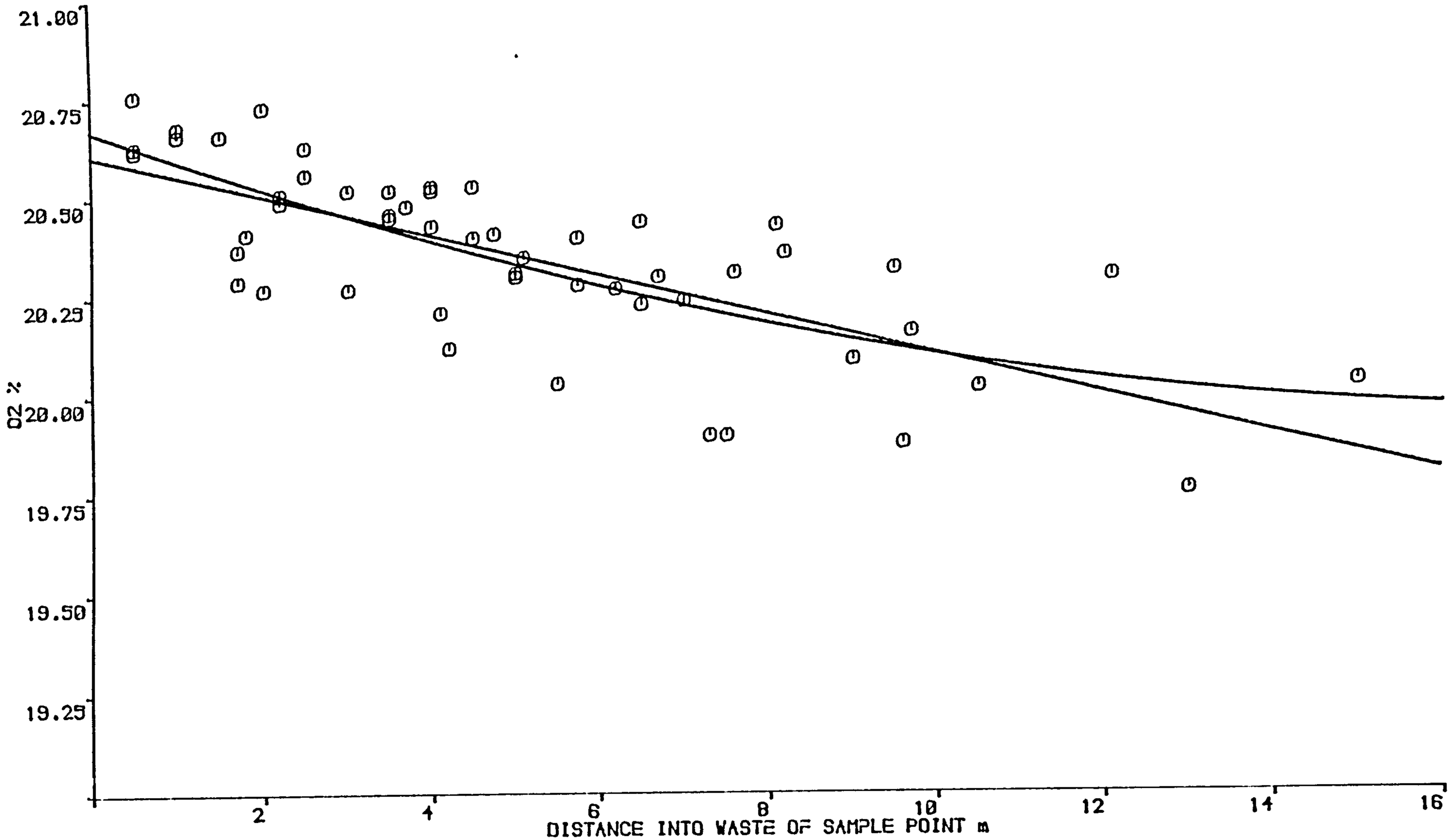


Figure 6.3 OXYGEN LEVEL vs DISTANCE INTO THE WASTE

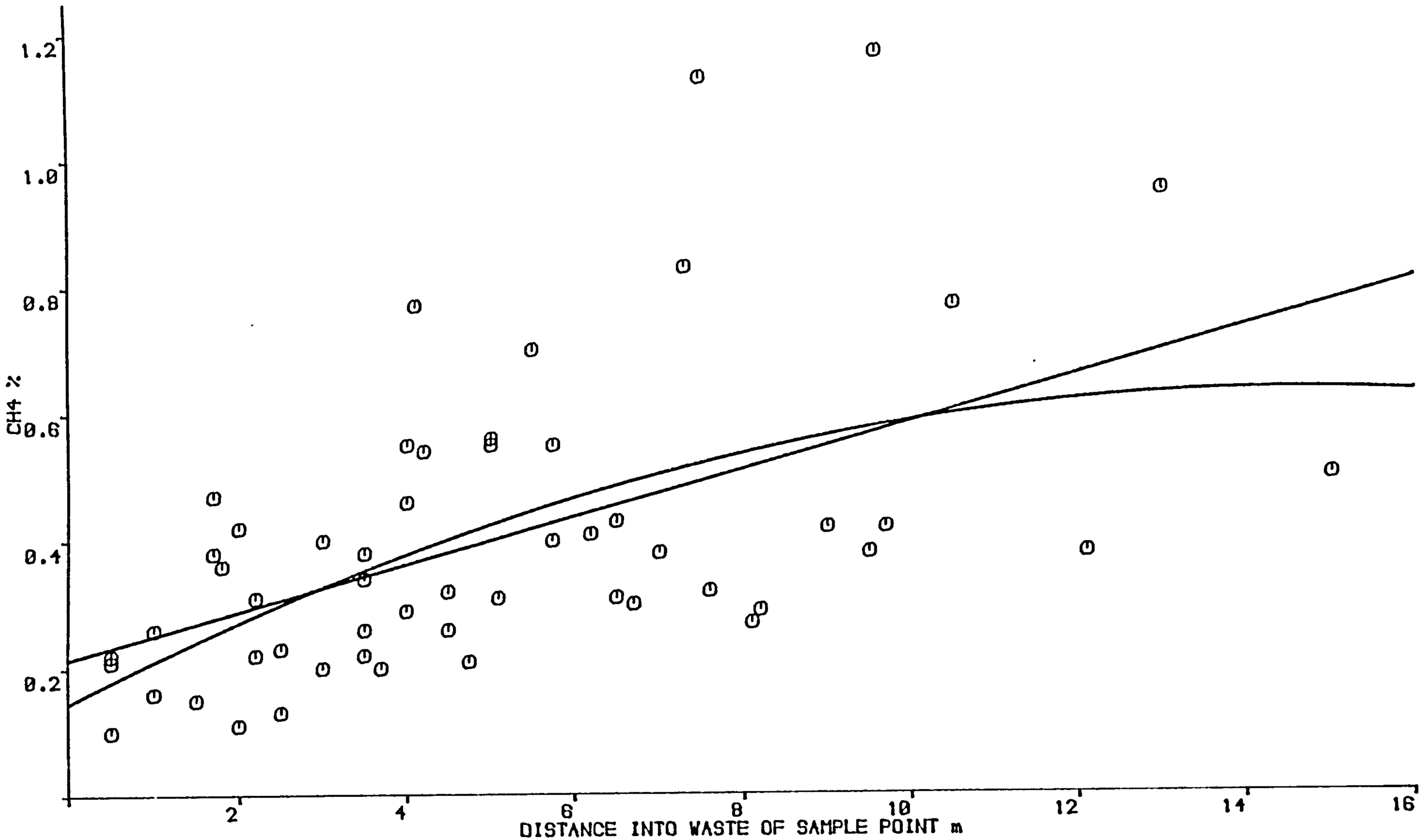


Figure 6.4 METHANE LEVEL vs DISTANCE

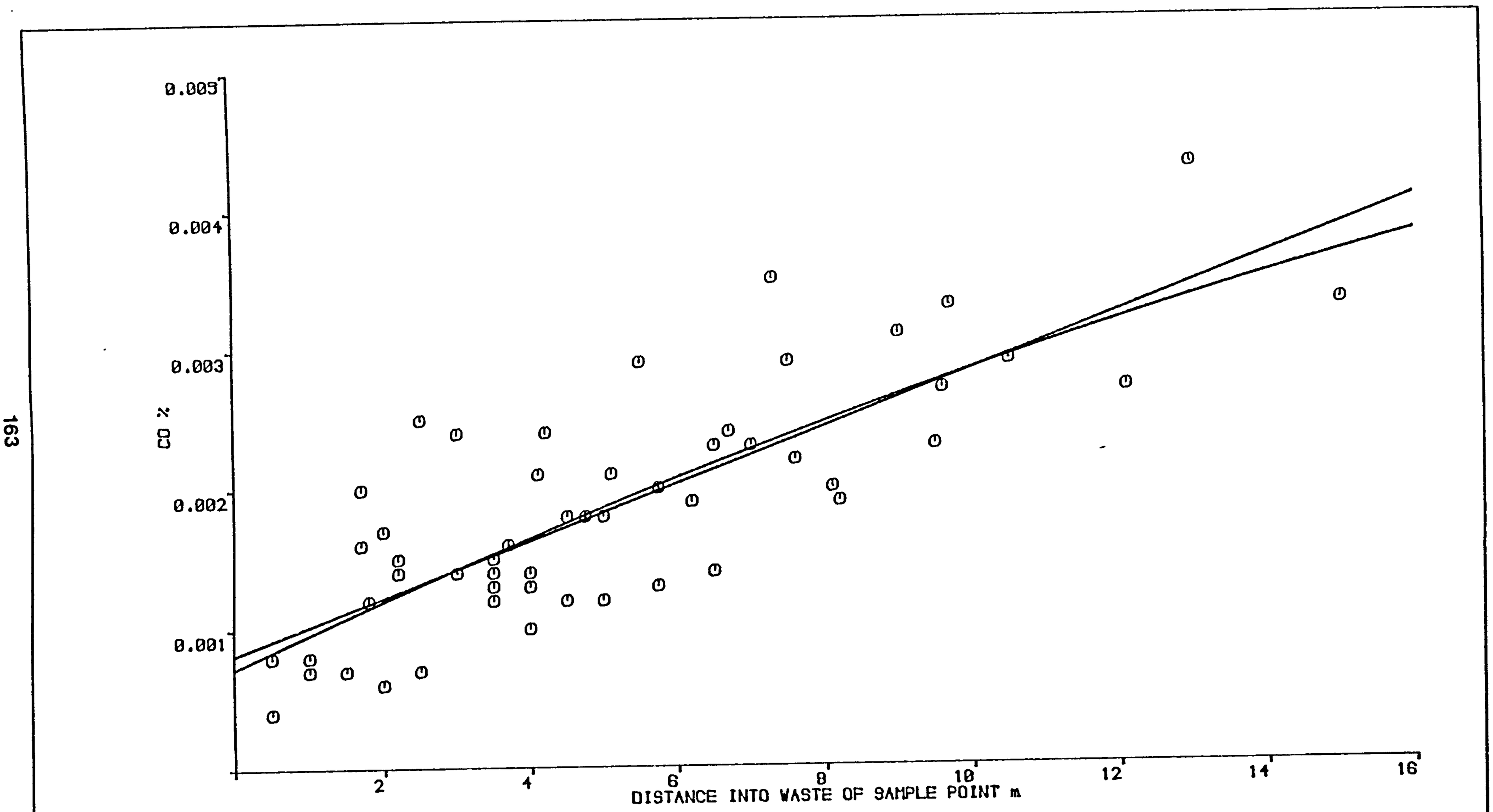


Fig. 6.5 CO LEVEL vs DISTANCE INTO WASTE

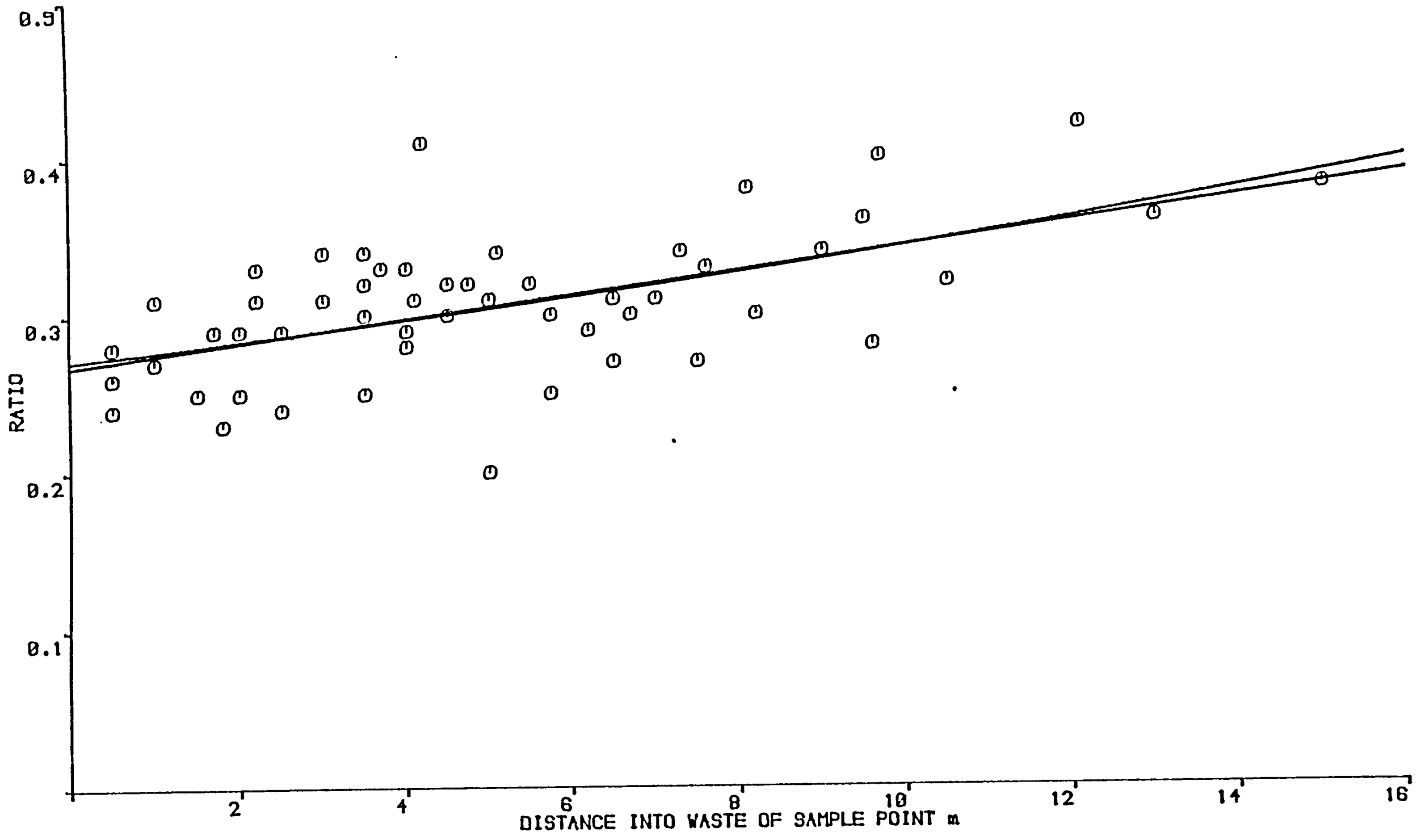


Fig. 6.6 GRAHAM RATIO vs DISTANCE

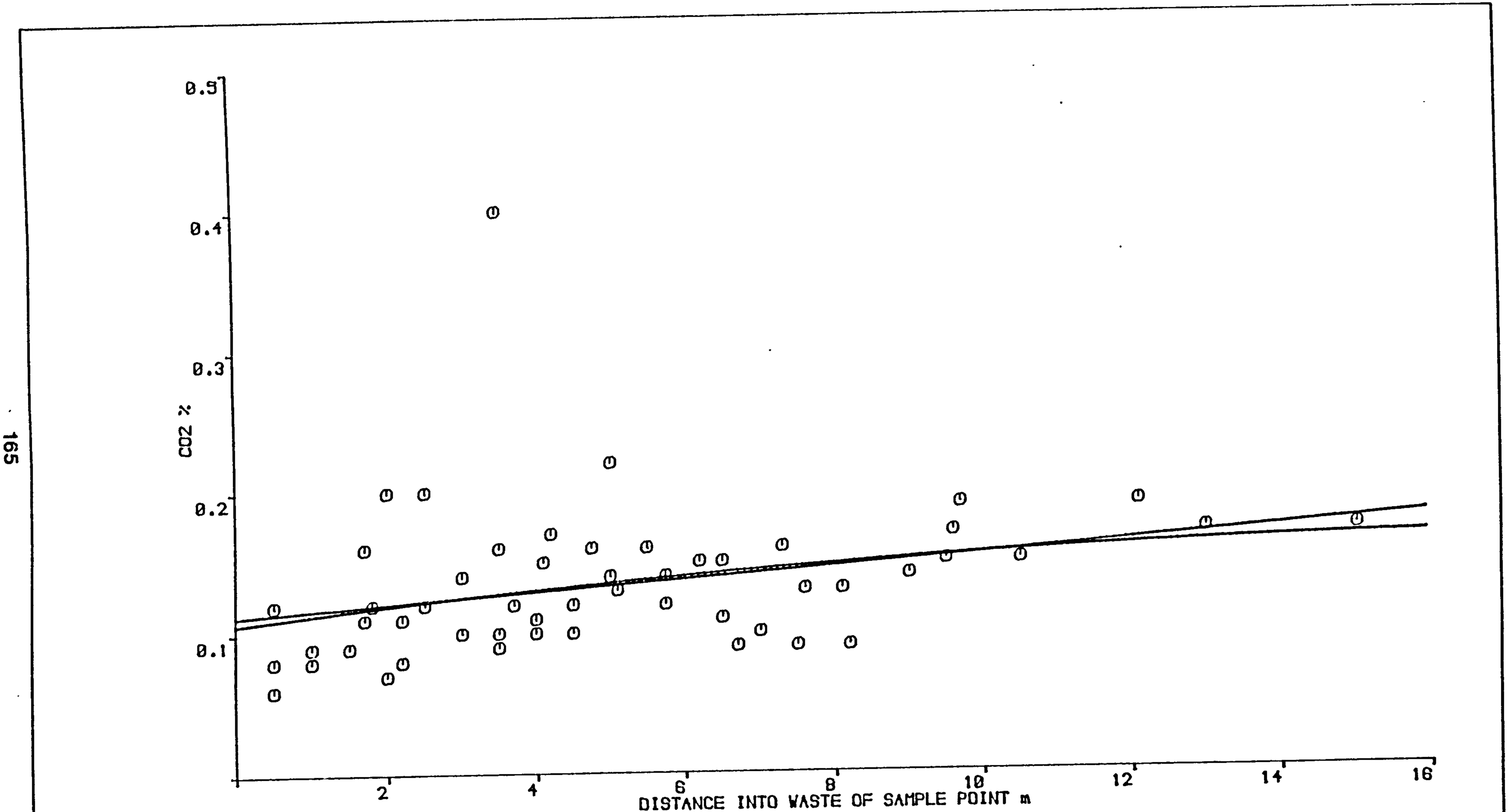
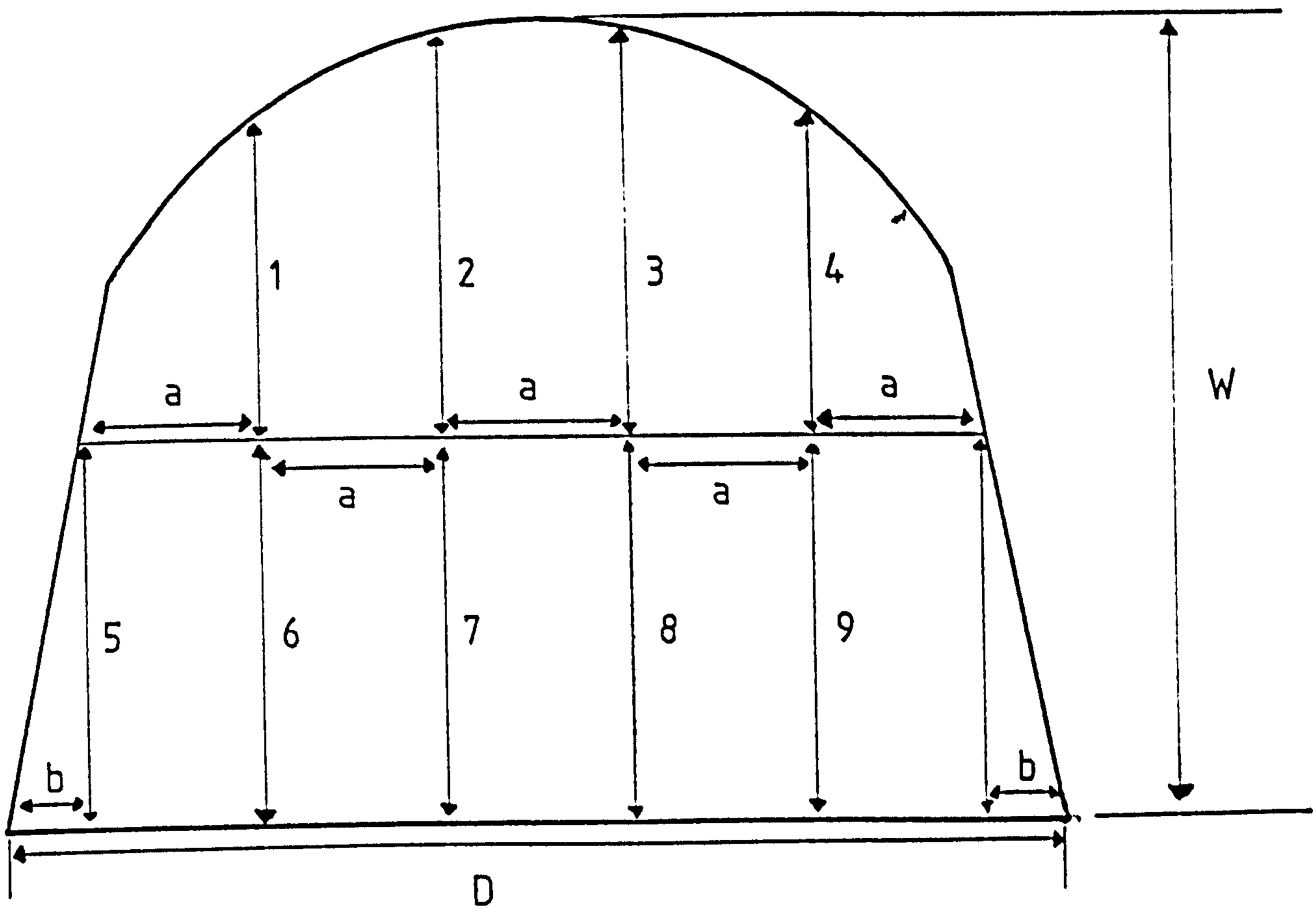


Fig. 6.7 CARBON DIOXIDE LEVEL vs DISTANCE

OBE RETURN



W = 12' 4" = 3.795m
D = 14' 8" = 4.463m
a = 2' 8" = 0.821m
b = 7" = 0.179m

1) 81cm
2) 113cm
3) 104cm
4) 78cm
5) 135cm
6) 135cm
7) 172cm
8) 180cm
9) 184cm
10) 184cm

Belt Area = 110 x 80 cm^2

Anemometer (50s)

50.0
48.0
53.0
46.0
49.0

Average = 49.2
Airspeed = 0.984m^2/s

Calculating from the above measurements:
roadway area = 10.185m^2

AIRFLOW = 10.02m^3/s

RETURN RIP

Arch size;
Height 12' = 3.692m
Width 16' = 4.923

Anemometer (50s)

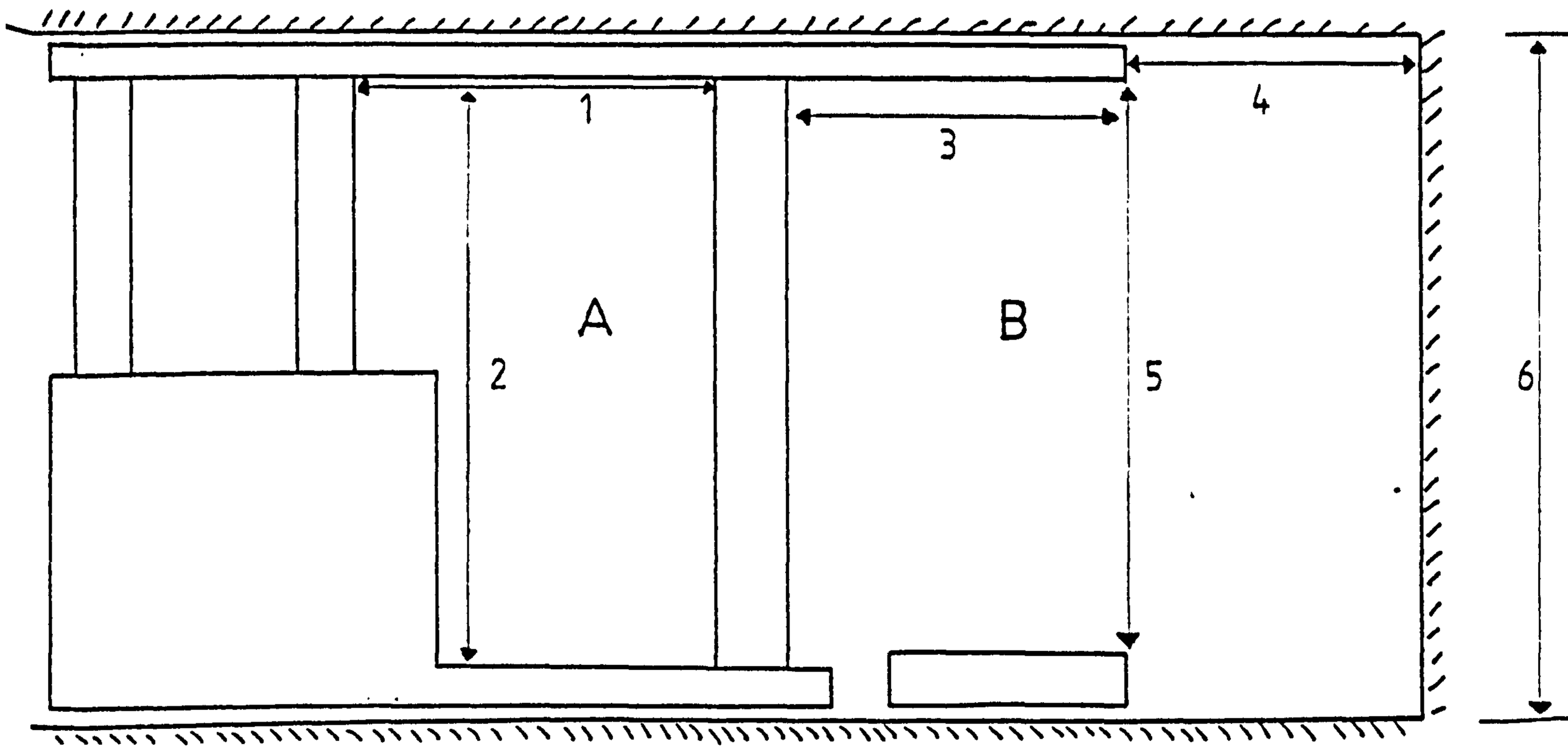
1) 33.0
2) 32.0
3) 33.5

Average = 32.83
Average Airspeed = 0.656m/s

Arch Area = 15.575m^2 (from NCB Ventilation Engineers Handbook)
allow 0.5m^2 for stage loader and other obstructions,
effective area = 15.075m2

AIRFLOW = 9.889m3/s

FACE LINE



DIMENSIONS (cm)

1) 90
2) 122
3) 107
4) 150
5) 150
6) 150
7) 170

ANEMOMETER (50s)

Chock Track
1) 20.0
2) 19.3
3) 19.0
Average Airspeed = 0.39m/s

Face Line
1) 113.0
2) 110.0
3) 112.0
Average Airspeed = 2.23m/s

AREAS

$A = 0.9 \times 1.22_2$
$= 1.098m^2$

$B = (1.07 \times 1.5)_2 + (1.5 \times 1.70)$
$= 4.155m^2$

Air Quantities

$A = 1.098 \times 0.39$
$= 0.428m^3/s$

$B = 4.155 \times 2_3 23$
$= 9.265m^3/s$

TOTAL AIRFLOW = $9.693m^3/s$

6.3.4 INTERPRETATION OF RESULTS.

The study was undertaken to investigate the change in gas levels with increasing distance into the waste. The possibility that changing barometric pressure could affect the composition of the waste gas was recognised.

The first set of results was taken during a period of stable barometric pressure. The general body samples did not change significantly in composition during the period. The two graphs, Figures 6.1 & 6.2 show what might be expected. The oxygen level fell, while pollutant gas levels rose. This would suggest slower airflow with increasing depth, and/or increased oxidation activity. The figure plotted at 220cm depth is an average of the 3 samples taken over the May holiday week. The main mine fan had been turned off for three days prior to the sample being taken on May 30th. June 3rd was the first working day after the holiday, and the sample taken at 270cm depth on June 4th was the first to be taken after the face had turned over following the holiday. The interruption of production and ventilation may well account for the anomaly in an otherwise smooth set of curves. Pickering (48) noted the effects of production and ventilation stoppages on methane emissions.

The first PVC sample tube was sealed off after 8 metres, and the more robust method described earlier adopted to sample gas from the waste. Again the composition of the general body air samples did not alter significantly during the time that samples were taken.

Over such a long period of sampling, the composition of the samples taken from the waste are certain to be affected by changing barometric pressure. By plotting all the points on one graph and drawing best fit curves, the overall effects of changes in barometric pressure are minimised. Scatter is worst on the methane plot, Figure 6.4, and it is known that barometric pressure changes greatly affect methane emission. This may further be affected by operation of the methane drainage system.

All the plots show some variation in the analyses, but within each of the scatter envelopes clear trends can be identified. The methane, carbon monoxide, Graham Ratio and carbon dioxide levels all rise. The Graham Ratio and carbon monoxide levels rise almost linearly with increasing depth into the waste. As expected the oxygen level falls. This is not a linear fall, but appears to be tailing off with increasing depth.

The sample pipe was laid along the floor. It is likely that there are convection effects in the waste, especially at the fringes where fresh, cold air can be drawn in. The analyses may indicate a lower level of pollutants, and a higher level of oxygen than exists further from the floor in the waste.

Despite this caveat there is evidence to show that oxidation was occuring at the fringes of the waste. There must have been substantial air movement at a depth of up to 15 metres in order to sustain the high oxygen levels observed. The airflow quantity measurements show that there was 5% greater airflow at the outbye end of the return than at the sample point on the face, and 2%

greater at the outbye end than at the return rip. The difference between the ripping lip and the face sample point was 2% of the airflow at the rip. The quantity of air flowing deep in the waste, and down beside the packs was not large. This shows the benefits gained from monolithic packing, when compared to figures given by Pickering (48) of not less than 10% difference between face airflow and that measured 60 metres from the return rip.

This small airflow of 5% of the air supplied to the district results in an atmosphere up to 15 metres into the waste sufficiently oxygen rich to sustain open fire.

Russian workers (52) have reported that to initiate spontaneous combustion, airflow velocities from 0.1 to 0.9 m/min, and an oxygen content of greater than 17% are required. Extrapolating the linear fall of the oxygen level, shows that the oxygen level in the waste may not fall below 17% until a depth of 70m from the face line.

The failure of the reinforced sample tube to reach a depth greater than 15 metres indicates that a different or much more robust method of sampling needs to be used. This was not possible with the resources availible.

6.4 CONCLUSIONS

This chapter has described the attempt made to identify the depth of airflow in the waste behind the face

The sampling program at Hucknall showed that a zone of airflow exists behind the chocks to a depth of at least 15 metres.

The oxygen level of the atmosphere in the waste falls, and pollutant levels rise, with increasing depth into the waste. Oxygen levels sufficient to sustain open fire exist to a depth of 15 metres.

In order to sample consistently from the waste, and to greater depths, more resources need to be directed to sampling methods to develop more robust methods. Such a method could be to lay the sample pipe through the pack, behind the chocks and parallel to the face, to the desired sample point. The pipe would be buried in fine coal, and strong steel piping used to protect the sample tube.

In addition to taking gas samples from the waste, the pressure difference between the sample point and the roadway needs to be measured to correlate the work in the next chapter with underground conditions.

When compared with previous measurements, monolithic packing reduces the air leakage though the waste.

It was decided that numerical modelling would provide a useful guide to the airflow patterns and quantities in the waste. The next chapter will examine the use of the finite element technique to predict pressure distributions and airflows in the waste, and the effects of nitrogen injection and faulting within the waste.

CHAPTER 7

FINITE ELEMENT MODELLING OF FLOW IN THE WASTE

7.1 INTRODUCTION

The waste behind a coalface consists of a large void filled with broken material. On an advancing face a pressure gradient exists across the waste, due to the difference in pressure between the intake and return roadways. This causes air to flow through the waste.

The leakage around a longwall district has been examined by other workers. The models most frequently used are very basic, calculating the quantity of loss from the intake, and gain to the return, at several discrete positions. Such information is used when leakage through the waste results in insufficient airflow at the face to combat pollutant gases, heat and dust, leading to rising ventilation power costs. Most ventilation network analyses ignore leakage, the volume of air circulating a district is assumed to be constant.

In areas where the environmental conditions are adverse, due to high gas emissions, high strata temperature or liability to spontaneous combustion, a knowledge of the pattern of gas flow through the waste could be used in measures to improve conditions.

A waste leakage model can be used in three areas:

1. Reduce the cost of ventilation.

2. Improve the accuracy of the ventilation network programs.

3. Guidance towards improving environmental conditions on the district.

Work in Russia (52) has identified air flow and air composition conditions likely to lead to spontaneous combustion. By suitable modelling it should be possible to identify where in the waste these airflow conditions may exist.

This chapter presents a model that was used to determine the pressure distribution in the waste. The method chosen was a relatively simple finite element computer program. The pressure distributions and air flow speeds produced by various permeability conditions in the waste, and pressure drops around the district were examined. The effects of faulting and nitrogen injection in the waste were examined.

7.2 STEADY STATE FLOW OF FLUIDS THROUGH POROUS MEDIA

A coalface waste is a porous medium. In order to successfully model the flow, the theory behind fluid flow through porous media needs to be understood.

Much of the work done on flow through porous media was based upon the theories of fluid flow through capillary tubes. Permeation through porous media is a more complex process than capillary flow. However, the equations governing a bundle of straight, parallel capillary tubes are used as the starting point for describing flow through a porous medium.

7.2.1 CAPILLARY FLOW

Five different regimes of flow have been identified in capillary tubes.

1. Poisseuille viscous flow
2. Molecular streaming
3. Molecular effusion
4. Orifice flow
5. Turbulent flow

Viscous flow occurs at low velocities in long tubes. Molecular streaming occurs in capillaries of very small diameter. Molecular effusion and orifice flow are special cases where flow is through short, narrow capillaries. At low pressures, effusion happens when the mean molecular path of flow is large when compared to the capillary diameter. As the pressure rises effusion becomes orifice flow, provided that the capillary is short and smooth enough to act as a nozzle. Typically these represent flow through fibres, such as paper.

7.2.1.1 Poisseuille Viscous Flow

This flow regime occurs in laminar flow of a viscous fluid. Laminar flow is characterised by a fixed set of streamlines within the tube. A fluid element traversing the same path at any given moment, as another previously, must follow the same path throughout its course.

An ideal viscous fluid adheres to the surface of a solid over which it is flowing. At the surface of the solid, the fluid velocity is zero. Similarly, at the wall of a capillary tube, the velocity is zero, and a parabolic velocity gradient between the centre of the capillary and the wall is created.

Poisseuille investigated the flow of incompressible fluids through a capillary, and found the relationship:

$$Q = \frac{-\pi R^4}{8\mu} \frac{\Delta P}{L} \qquad (7.1)$$

Q : Volume flow rate (m^3/s)
R : Radius of capillary (m)
μ : Fluid viscosity (Ns/m^2)
ΔP : Pressure drop (N/m^2)
L : Length of tube (m)

However, not all fluids are incompressible. If a compressible fluid is used, then:

$$Q_2\rho_2 = Q\rho = \frac{-\pi R^4}{8\mu} \frac{dP}{dx} \rho \qquad (7.2)$$

End condition Intermediate condition

ρ : Fluid density (Kg/m^3)
dP/dx : Pressure gradient

If the fluid is treated as an ideal gas, under isothermal conditions density is a function of pressure, $\rho = CP$ (C = constant)

$$Q_2P_2 = \frac{-\pi R^4}{8\mu} \frac{dP}{dx} P$$

Separating and integrating:

$$Q_2P_2L = \frac{-\pi R^4}{8\mu} \frac{P_2^2 - P_1^2}{2}$$

$$Q_2 = \frac{-\pi R^4}{8\mu L} \frac{(P_2+P_1)(P_2-P_1)}{2P_2} \qquad (7.3)$$

If $\frac{(P_2+P_1)}{2}$ = Mean pressure = P_m

P_2 = Mean pressure
Q_2 = Mean quantity, Q_m

Then

$$Q_m = \frac{-\pi R^4}{8\mu} \frac{\Delta P}{L} \qquad (7.1)$$

7.2.1.2 Molecular Streaming

For Poisseuille viscous flow to occur, the diameter of the capillary must be much greater than the mean molecular path of the fluid molecules. When this is not the case, experimental flow results give a higher flow than is predicted by 7.3. This arises because of the fact that the fluid velocity at the capillary walls is not zero, an assumption made in Darcy's equations. The fluid slips past the walls. The quantity of gas flowing appears greater than the diameter would warrant using Darcy's equation. It is unlikely that this flow regime will occur in a coalface waste, and will not be further discussed here.

7.2.1.3 Turbulent Flow

At high flow velocities, the laminar flow breaks down. This change is a function of the tube diameter, flow velocity, fluid viscosity, and density. Beyond a transition point, flow ceases to be wholly laminar. The streamlines break down, and random eddies are formed in the flow. The flow gradually becomes wholly turbulent as the velocity rises. Poisseuille's formulae no longer apply when the flow is not wholly laminar.

Circular straight tubes are dynamically similar if the Reynold's number, a dimensionless coefficient is the same:

$$Re = \frac{2\rho R v}{\mu} \qquad (7.4)$$

v : Fluid velocity (m/s)

Above a value of Re = 2200, flow in straight capillaries is no longer laminar. In practice, a transition zone exists in which the flow is neither wholly laminar nor turbulent. The value of Re represents the lower end of this range.

7.2.2 FLOW IN POROUS MEDIA

Poisseuille's equations can be used as a starting point to describe the flow through a porous medium. If the medium can be considered to be a bundle of straight, parallel tubes, then the flow will be:

$$Q_t = -n\frac{\pi R^4}{8\mu}\frac{\Delta P}{L} \qquad (7.5)$$

Q_t : Total flow rate (m^3/s)
n : Number of tubes

$$Q_t = -\sum_{j=1}^{k} n_j \frac{\pi R^4}{8\mu}\frac{\Delta P}{L} \qquad (7.6)$$

n_j : Number of tubes of radius R_j
k : Number of tube groups of different radii

The value of $\pi/8 \sum n_j R^4$ can be treated as a constant for a particular group of tubes, so:

$$Q_t = C\frac{\Delta P}{\mu L} \qquad (7.7)$$

However the fluid conducting channels in a porous medium consist of many interconnected tubes of different radii. For anything but the most geometrically simple porous media, some other means of assessing the flow coefficient of rock is needed.

D'Arcy investigated the flow of water through sand, and found the empirical relationship:

$$Q = KA\frac{\Delta h}{L} \qquad (7.8)$$

Q : Flow rate
K : Fluid flow coefficient
A : Area of flow
L : Length of flow
Δh : Difference in head of water

Further investigation showed that this equation could be extended to other fluids and rocks. The constant K is written as k'/μ. The permeability, k', is a property of the rock alone, and, μ, the dynamic viscosity of the fluid.

In SI units,

$$Q = \frac{Ak'\Delta P}{\mu L} \qquad (7.9)$$

Q : Flow rate (m^3/s)
k' : Permeability of the rock (m^2)
μ : Dynamic viscosity of the fluid (Ns/m^2
A : Area of flow (m^2)
ΔP : Change in pressure (N/m^2)
L : Length of flow (m)

This applies to incompressible fluids. A more general relationship exists for compressible fluids, but over small pressure differences most gases can be treated as incompressible.

The Darcy equation is applicable only when the flow is in the laminar regime. The Reynolds number for flow through a porous medium is given by:

$$Re = \frac{\rho D Q}{\mu A} \qquad (7.10)$$

D : Average particle diameter
Q : Volume flow rate
A : Area of flow
ρ : Fluid density
μ : Dynamic viscosity

The flow is said to be wholly laminar for Reynolds numbers of less than 1, with a transition to turbulent flow between 1 and 10.

7.2.2.1 Turbulent Flow Through Porous Media.

For Darcy's equation to be applicable, the fluid flow must be in the laminar regime. Resistance to flow arises from the viscosity of the fluid, and consequent friction between the particles of fluid and the medium.

At higher rates of flow, a non linear laminar flow regime is entered. The streamlines become distorted, and stable local turbulence forms around the particles. The pressure drop in this regime can be expressed as: (53)

$$P = (A + BQ)Q$$

where A and B are constants, dependent on the fluid, permeability of the medium, and velocity of flow in the medium.

As the flow rate rises further, highly unstable local turbulence is developed around the particles of the medium. The constants A and B change in value.

The varied size of the particles and flow paths means that there is no clear transition point between linear laminar flow and turbulent flow, but results in a gradual transition. Mixed flow regimes exist in a porous media at higher flow rates.

It is possible that fully turbulent flow is never reached, due to the fluid friction and the packing of the particles. At Re values of above 500, the pressure drop can be approximated by $P = CQ^2$, where C is a constant (54).

Extensive research has been done on non-Darcy flow in porous media. There is no universal equation applicable over a wide range of Reynolds numbers. At lower Reynolds numbers a satisfactory flow approximation can be obtained by applying Darcy's equation with an increased resistance factor.

7.3 WASTE FLOW MODELLING TECHNIQUES

The problem when modelling the air leakage around a longwall district has been to combine ventilation network programs with methods of modelling that yield useful information, while

remaining efficient users of computer time.

Most ventilation planning requires an estimate of the losses around a district. The type of modelling system most easily combined with a network program is similar to that in Figure 7.1. Airways of representative resistance lead from points in the roadways to a central node in the waste. This gives an estimate of the losses to the waste, the quantity flowing on the face, and the power required to ventilate the district. It does not yield any information of the position of leakage along the roadway. To obtain an estimate of the fall in air quantity along the intake roadway Stokes (50) divided the waste into representative airways that run parallel to the face line, Figure 7.2 This method obtains the magnitude of the air loss from the intake, but gives no information about the airflow within the waste. To do this, the airflow cannot be constrained by representative airways.

Brunner (55) divided the waste up into rectangular elements, Figure 7.3a. The elements were assigned permeabilities and flow areas to correspond with the redistributed vertical stress over the caved waste. Airways were constructed diagonally through each element, Figure 7.3b, and airway resistance values calculated from the element permeability and area. Pressure values were assigned around the waste. A modified ventilation network program was used to calculate pressure drops and airflows in the airways. This dense mesh of airways gave a good idea of the airflow pattern in the waste, and a simplified model with fewer airways was constructed.

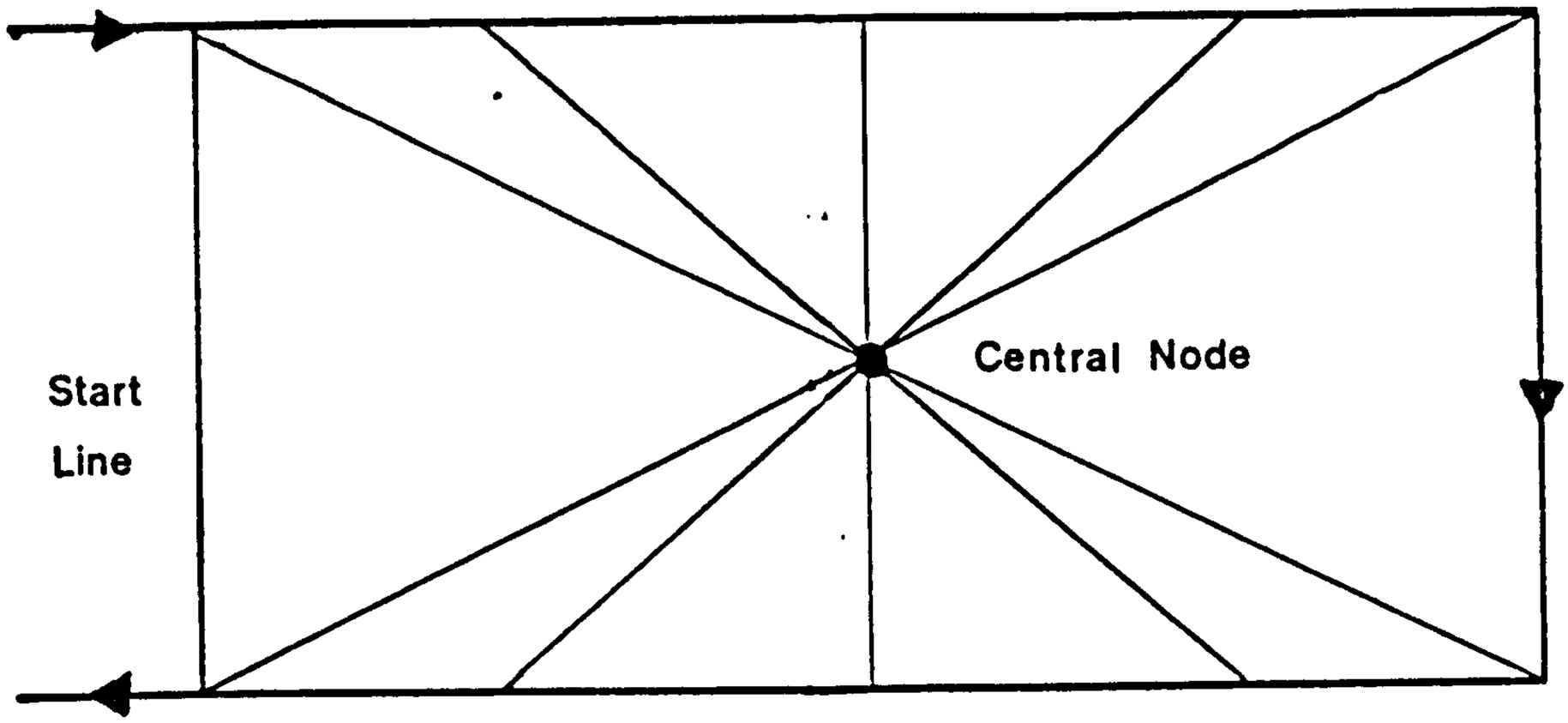


Figure 7.1 Central Node and Airway Method of Goaf Leakage Modelling

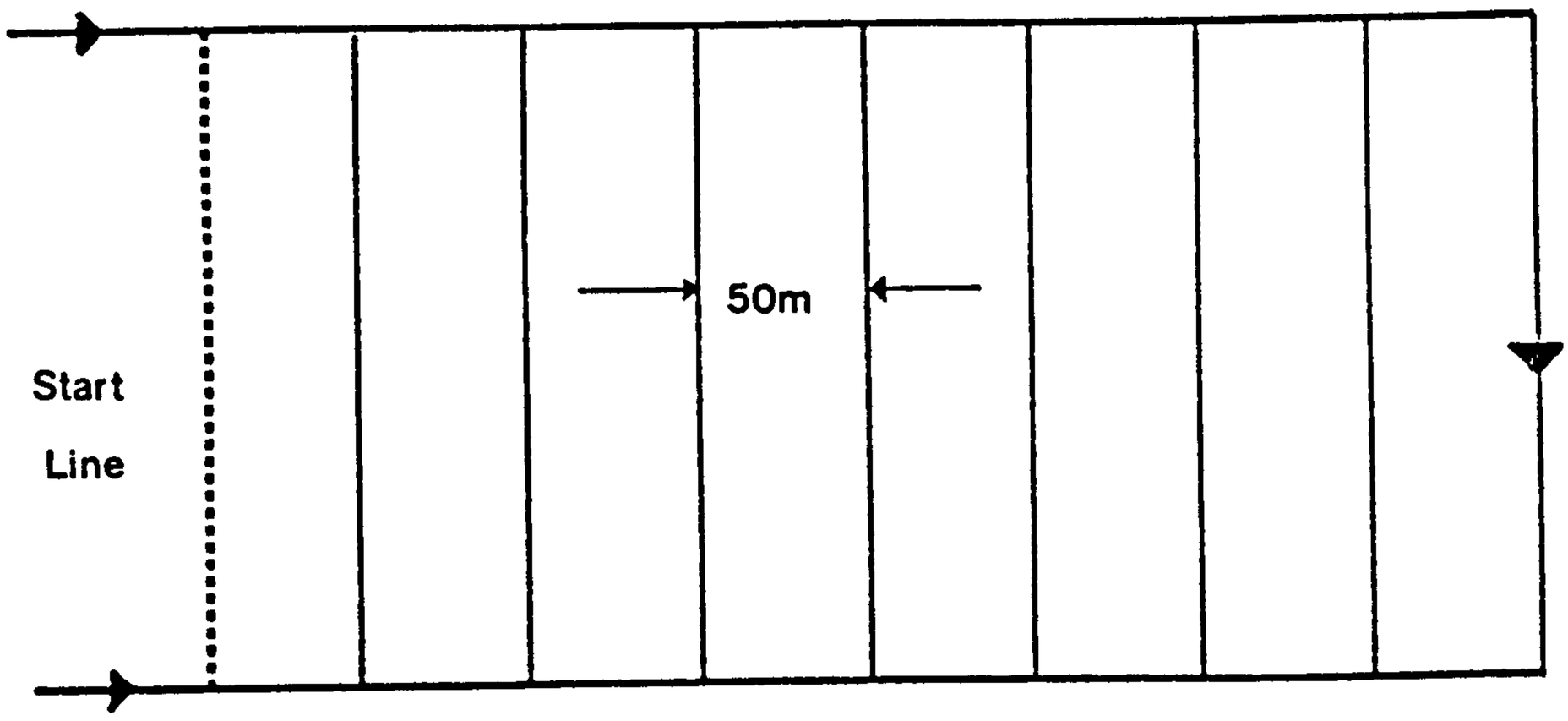


Figure 7.2 Parallel Representative Roadway Method (50)

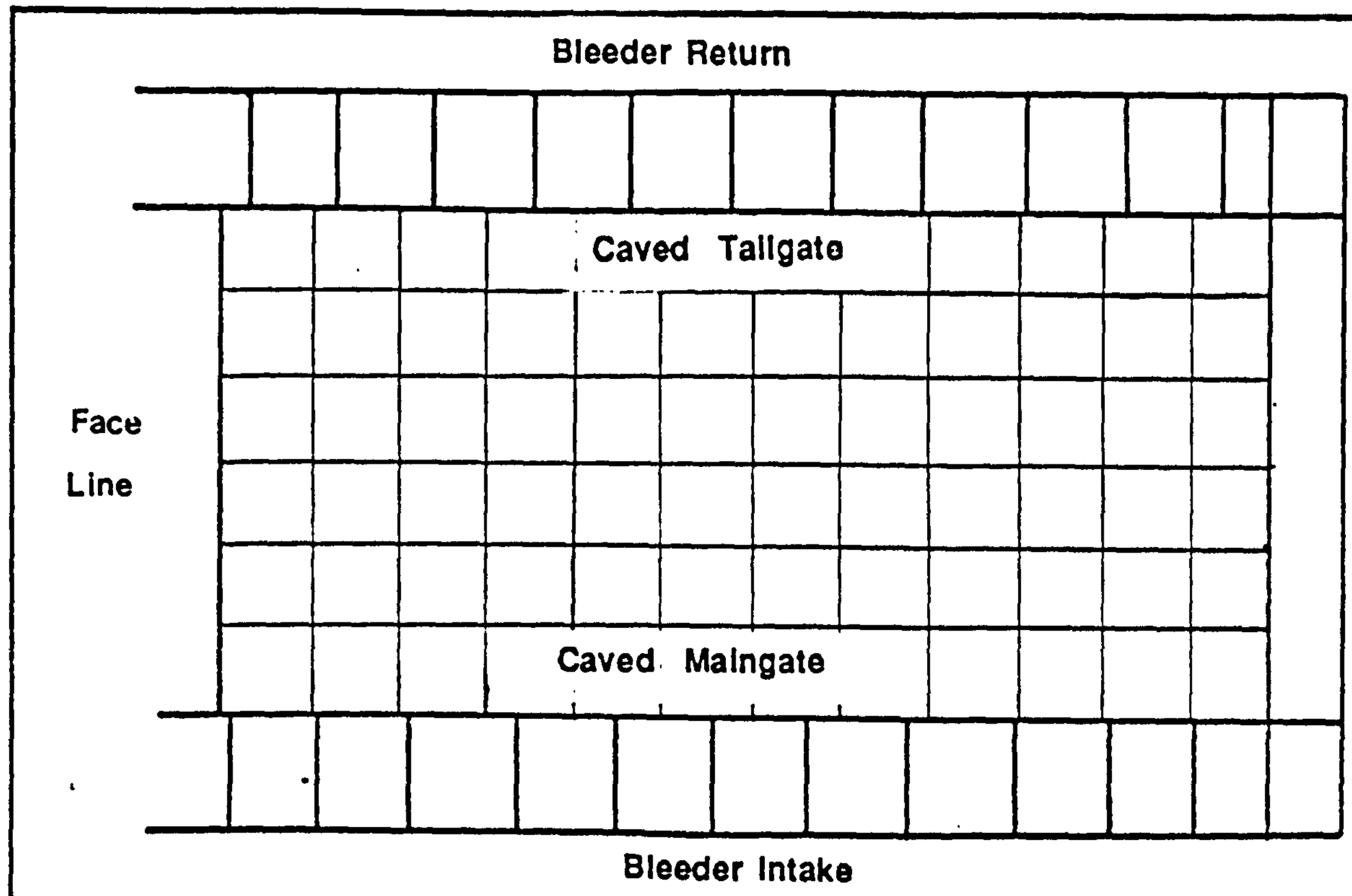


Figure 7.3a Rectangular Resistance Elements

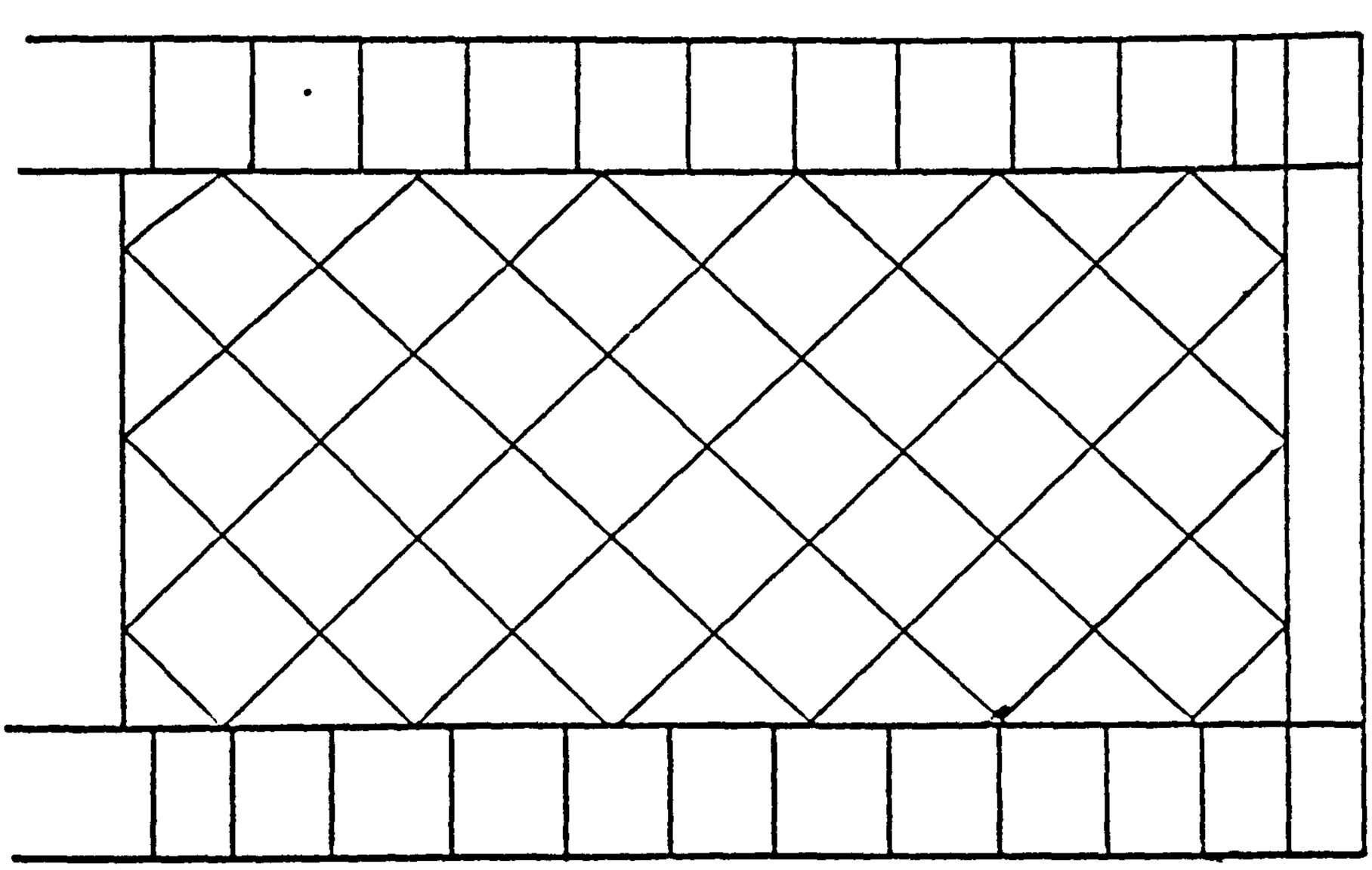


Figure 7.3b Representative Airways Constructed Through Elements

Figure 7.3 Modelling of Waste Airflow Using a Mesh Method (55)

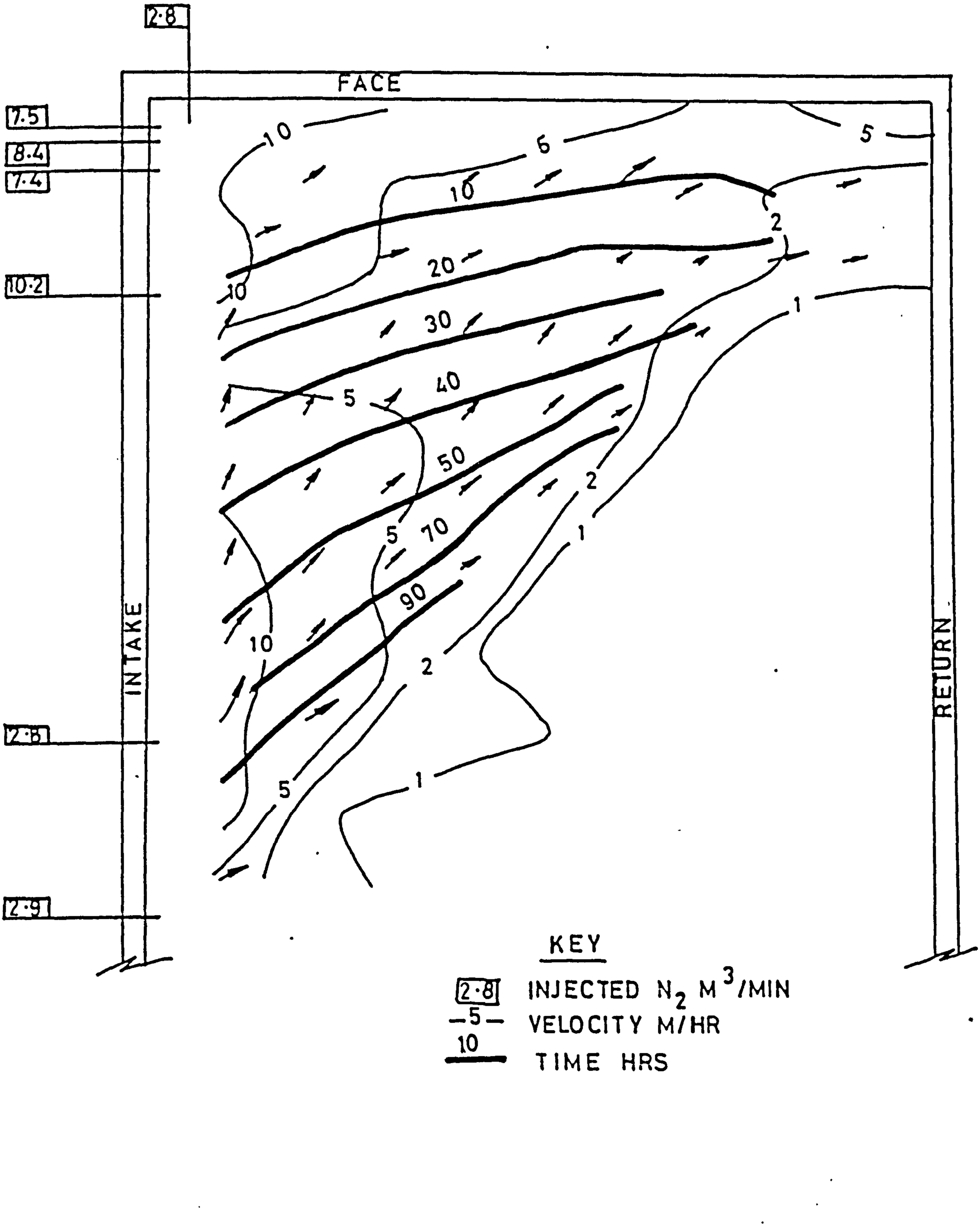
Figure 7.4 shows the diagram that was used by the South Midlands area of British Coal to model waste flow under nitrogen injection conditions. No details of the model, other than that it is computer generated, have been published (30). Mention has been made earlier (Chapter 3.3.3) of proposed airflow and oxidation zones in a longwall waste.

To calculate the pressure distribution and resulting airflow paths in the waste, the area cannot be modeled using representative airways. The method must treat the area as a continuum.

An alternative to numerical modelling, usually computer based, is to solve the governing Laplace differential equation using calculus. This has been done in Russia (56). An expression is obtained for the pressure in the waste at any point, as are expressions for the x and y components of velocity. By integration an exact equation for the quantity of air lost to the waste is obtained. In order for this analytical technique to remain usefully simple, a uniform permeability in the waste has to be assumed.

The finite element method solves the differential equations governing flow in the waste, subject to boundary conditions. It allows the area to be modelled as a continuum. A computer has to be used to enable the method to be efficiently applied.

Figure 7.4 Nitrogen Goaf Flow Simulation (30)



7.4 MODELLING WASTE FLOW USING THE FINITE ELEMENT METHOD

The finite element method is a numerical procedure for solving the differential equations encountered in engineering. The method uses several different mathematical concepts to produce a system of linear equations, which are solved to give values at discrete points in the area of interest. Many text books and papers have been published on the method. The core of the computer program used to model the waste flow was taken from Segerlind (57), and subsequently altered.

The area of interest is divided up into elements. These may be of almost any shape, but as the complexity of shape increases, so does the complexity of the computer program required. At the corners of each element there are nodes that define the shape of the element. More complex programs allow nodes along the element sides. Each element is assigned a material coefficient value, permeability in this case. The nodes along the perimeter of the area are given known boundary condition values. It is possible to apply derivative boundary conditions, and to have point sources of flow within the mesh. Using the data supplied the program calculates the value of the variable at each node in the mesh, and the value of the gradient in the x and y directions. Thus in the waste modelling case, the air pressure in the waste is calculated at each node, as are the x and y velocities.

Graphical subroutines were written to draw the mesh, and plot pressure contours in the mesh. The resultant airflow velocity was calculated from the x and y velocities, and velocity contours plotted. The velocity contour plots are not streamline plots, and do not indicate airflow direction, only speed. The direction of the airflow can be inferred from the pressure contour plots, as the direction of flow is at right angles to the pressure contours.

Laminar flow conditions were assumed to exist throughout the area. This was done to simplify the finite element model. As will be shown later, this assumption may not hold true under some conditions. The program was not changed as laminar flow breaks down only in limited areas, and the model provides a good approximation to pressure distribution and flow in the waste.

7.4.1 INPUT DATA FOR THE FINITE ELEMENT CALCULATIONS

The input data required for the program consists of:

1. The co-ordinate and element mesh of the area
2. Permeabilities of the elements
3. Boundary pressure data
4. Point sources of flow, to simulate nitrogen injection if applicable

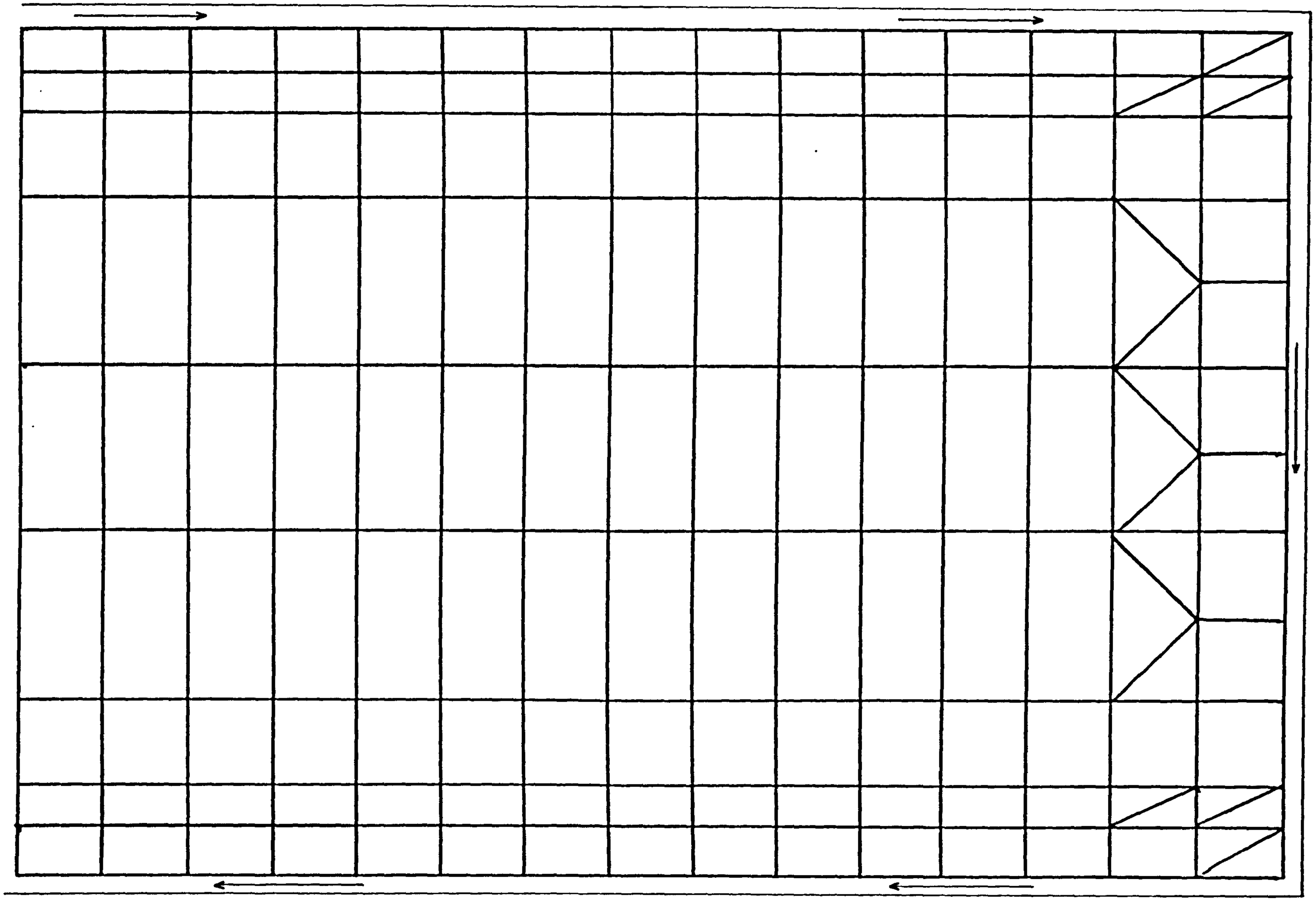
7.4.1.1 Element Mesh Of The Area Modelled

The areas at risk from spontaneous combustion are the face start line, and the zone immediately behind the face. Heatings on the start line can be relatively easily isolated and combatted. Heatings behind the face line are not so easy to isolate. The area modelled in the first set of exercises represents a 210m long advance face, with 300m of roadway leading to it. The area in the second set of exercises is a 250m long face, with 400m of roadway leading to it. To obtain greater accuracy of solution in certain areas, such as the face ends, a greater density of nodes has been used. Along the roadway sides, narrow elements were used to simulate the pack and the unconsolidated waste corridor beside the pack. At the face ends, and along the face, small elements are used, while in the centre of the waste, where accuracy is not vital, larger elements are used. The basic element mesh is shown in Figure 7.5

7.4.1.2 Element Permeability

The major assumption that has to be made is that of the element permeability. Insufficient work has been published on the gas permeability of broken materials under load to assign accurate permeabilities to the elements. The redistributed stress field over a coal face waste is dome shaped (58). With the increased load, and consolidation, at the centre of the waste, the permeability will be lower at the centre than at the edges.

Figure 7.5 Element Mesh Used in Finite Element Calculations



The permeability of the area around the roadways will mostly arise from breaks in the strata around the pack. The monolithic pack will have a low permeability as it is solid. At Harworth, where the pressure measurements were taken, the pack is mechanically stowed by a cam packing system using the debris from the Dosco ripping machine. Initially these packs may have a high permeability, but they are made from a wet shale slurry. This sets to form a pack that is so strong and solid that to make refuges in the roadway sides explosive charges are used. It is the author's opinion that this kind of pack will have a permeability of a similar order of magnitude to a monolithic pack.

Three exercises using different graduations of permeability within the waste were done. In the first, a constant permeability throughout the waste was used. The next step was to divide the waste into different areas, and assign permeabilities to each area. The areas chosen, and relative permeabilities were:

Centre of the waste	:	Low permeability
Waste perimeter	:	High permeability
Roadside pack	:	Medium permeability
Elsewhere in the waste	:	Medium permeability

This is referred to later as the Varying Permeability model.

A further step was to use a more gradual fall in permeability from the waste perimeter to the centre of the waste. This is referred to as the Graduated Permeability model.

An important area when considering outbreaks of spontaneous combustion is that around faults and discontinuities in the strata. The breakup of the strata decreases the resistance to airflow, while there is a large volume of potentially active material left along the line of the fault. The most critical line of a fault has been found to be one that runs from the intake roadway to the face. To model a fault in the waste, the element permeability was increased along an imaginary fault line.

The permeability values used ranged from 10^{-6} m^2 TO 10^{-8} m^2. These are comparable with those used by other workers (55,56).

7.4.1.3 Boundary Pressure

Two sets of boundary pressures were taken on different working faces, both at Harworth Colliery in North Nottinghamshire.

The first set was taken on Harworth 1's, in January 1985. At the time the pressures were taken, the face had advanced 1500m, it was 210m long with 210 powered supports on it. Pressure readings were taken using a micro aneroid barometer, giving readings of static pressure in inches of mercury. The pressure was measured at several points around the district, shown in Figure 7.6, the values are recorded in Table 7.1. There was a surface pressure drop of less than 0.0051 KPa during the time that readings were being taken underground. These pressure values were then interpolated to give boundary pressures on the nodes at the perimeter of the mesh.

Measurement Point	Pressure Ins. Hg	KPa
Stn 1	33.140	112.225
Stn 2	33.109	112.120
200 Chock	33.110	112.124
150 Chock	33.102	112.097
100 Chock	33.102	112.097
50 Chock	33.099	112.086
20 Chock	33.100	112.090
Stn 3	33.100	112.090
Stn 4	32.998	111.744

Table 7.1 Static Pressure Measurements, Harworth 1's

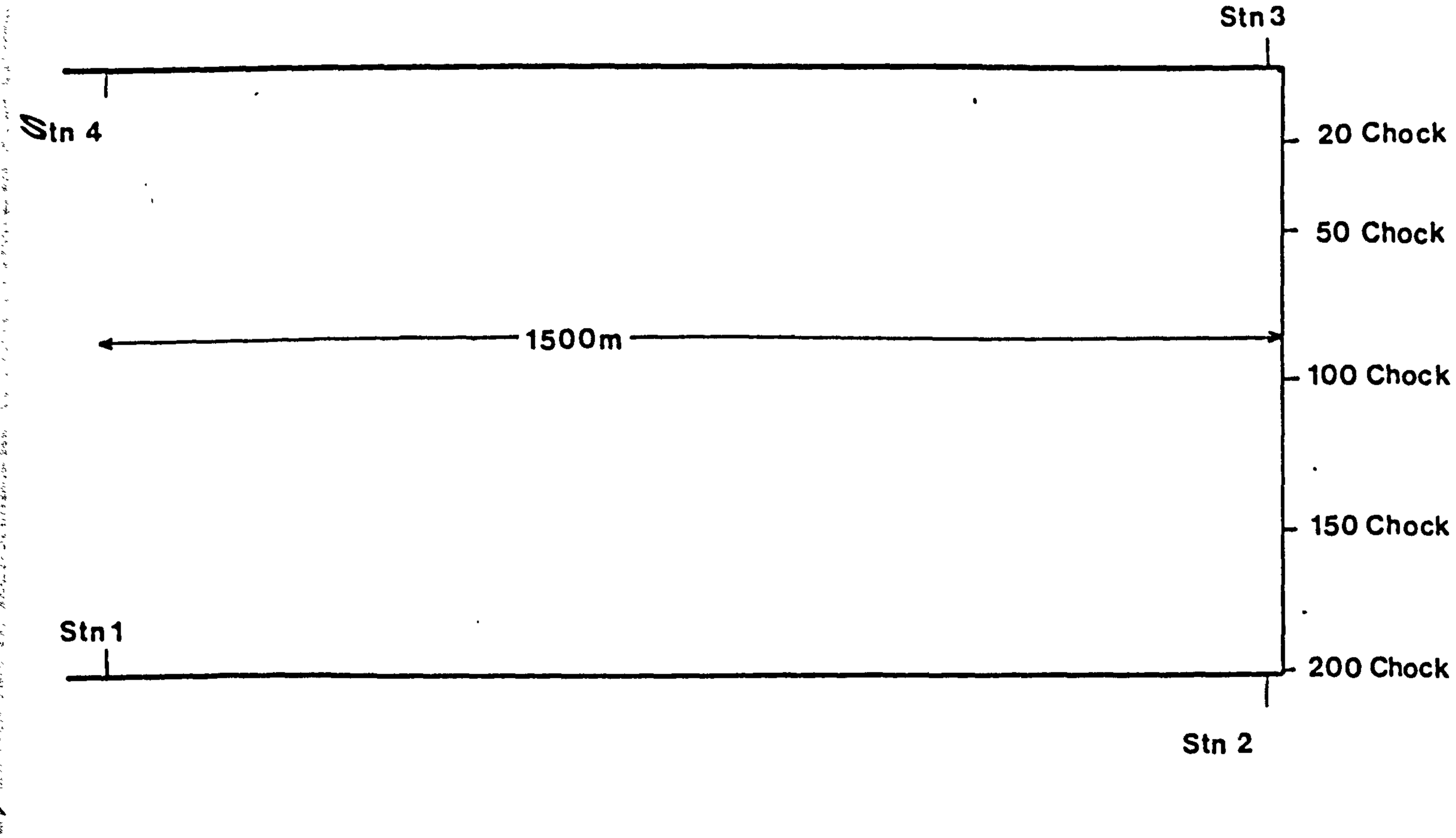


Figure 7.6 Static Pressure Measurement Points, Harworth 1's

The second set of readings was taken on Harworth 3's in March 1987. This face had advanced 2000m and was 250m long, had 160 chocks on it, with an extraction height of 1.8m. The pressure drop was measured using the gauge and tube method. The distance between each measurement point was approximately 90m in the roadways, and 75m along the face line, as shown in Figure 7.7. The results are in Table 7.2. The pressure at the furthest point outbye in the return was taken as a datum, the other values are pressures above this datum.

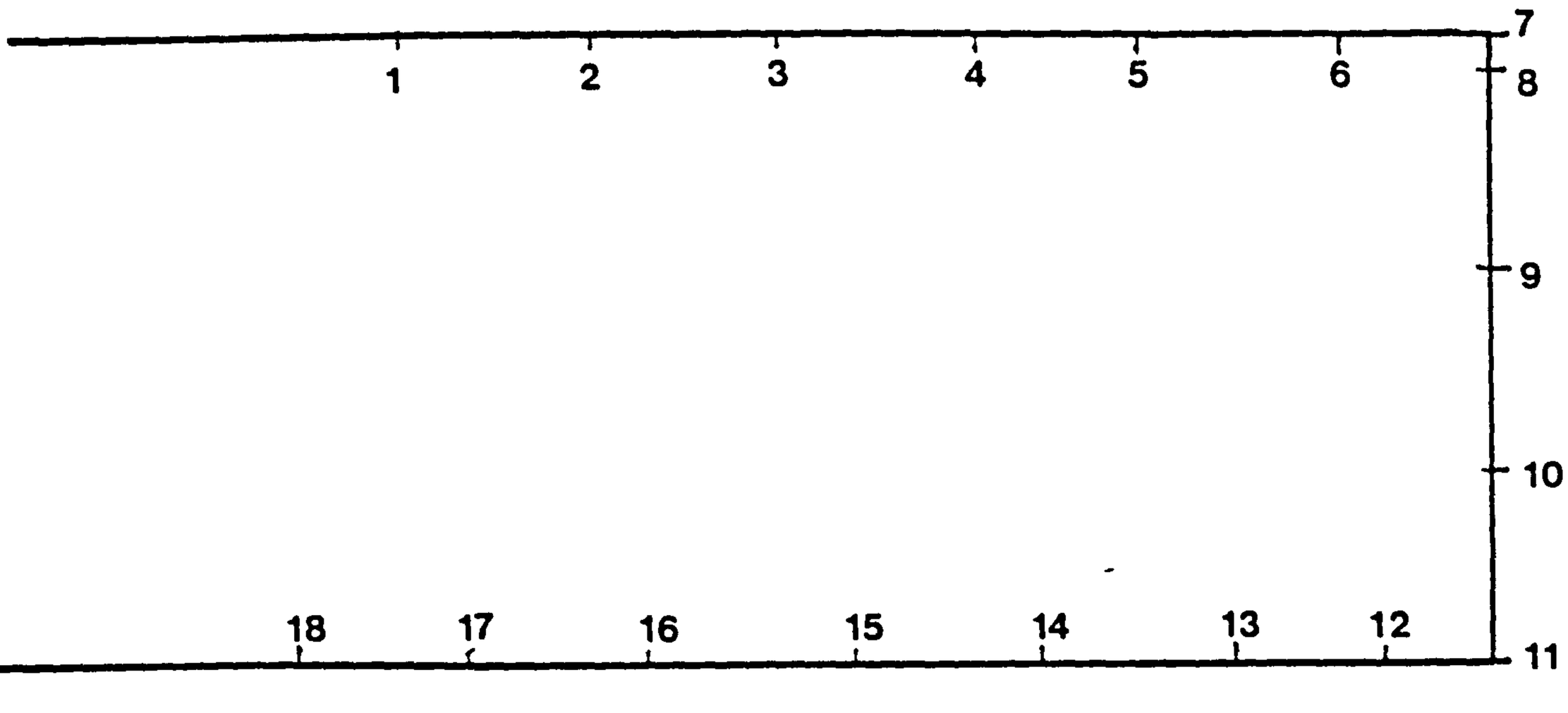


Figure 7.7 Pressure Measurement Points, Harworth Three's

STATION	POSITION Metre mark/ Chock No.	PRESSURE Pa	AIRFLOW m^3/s	COMMENTS
1	1475m	320		
2	1570m	314		
3	1668m	306		
4	1767m	293		
5	1850m	288		Roadway dint
6	1950m	282		
7	2028m/ Face end	267	22.23	
8	150 Chock	257		Face end & ripping lip
9	100 Chock	234		
10	50 Chock	210		
11	Face end/ 1954m	142	22.25	Shearer
12	1900m	120		Face end & ripping lip
13	1823m	85		
14	1725m	69		
15	1632m	52	23.3	
16	1525m	24		Roadway dint
17	1435m	18		
18	1350m	0	23.39	

Table 7.2 Harworth 3's Pressure Survey

At the same time as this pressure survey, airflow quantity measurements were taken, using the same method as in Chapter 6.

Both sets of pressure measurements were taken on non-working shifts, to avoid the pressure and quantity around the district fluctuating due to personnel and material movements around the mine.

7.4.1.4 Nitrogen Injection

The quantities used when modelling the pressure distribution under nitrogen injection conditions were taken from the model by Lord (30), Figure 7.4. The faces in the South Midlands area, where this simulation comes from, have much lower ventilation pressures than are found at Harworth, as ventilating quantities are lower and the faces have a greater height of extraction. Typically the pressure difference between intake and return in this area is less than 100Pa (59).

7.4.2 EXERCISES USING THE FINITE ELEMENT METHOD

A listing of the program used, and examples of input and output files are given in Appendix 2.

7.4.2.1 Harworth 1's

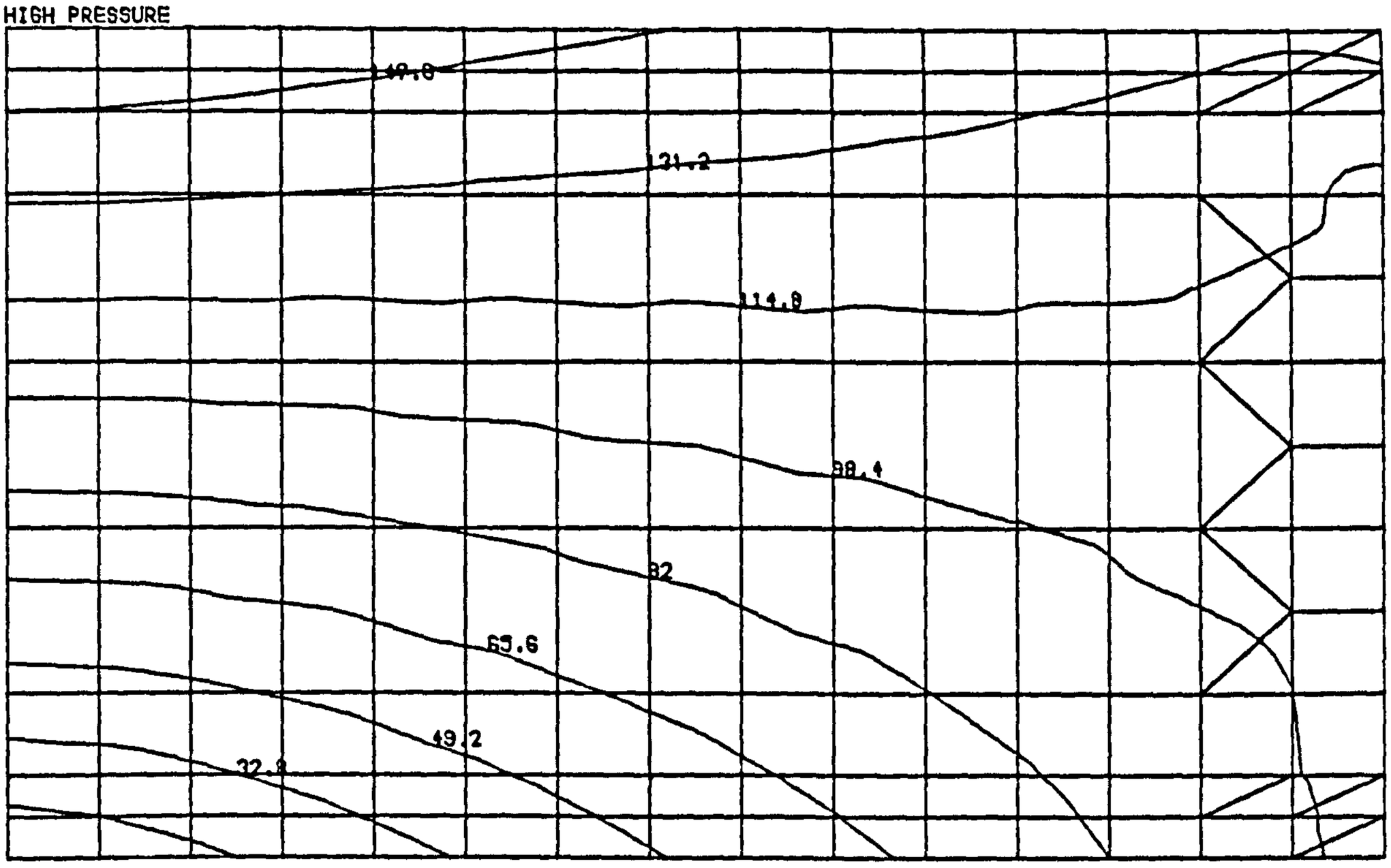
7.4.2.1.1 Constant Permeability

The pressure distribution resulting when a uniform permeability throughout the waste is assumed is shown in Figure 7.8a. As might be expected, the pressure drop across the waste is regular. The shape of the contours towards the edge of the mesh is governed by the pressure drop in the roadways. Air flows at right angles to the pressure contours. It can be seen that the general trend of the airflow will be straight across the waste. The velocity contour plot, Figure 7.8b, shows that the velocity is highest furthest away from the face line, where the pressure gradient is the greatest.

Stokes (50) found that leakage in a very unconsolidated waste was parallel to the face line, while the rate of increase in quantity leaking fell as the face line was approached. Highton (32) found that heatings could be combatted by sealing in the intake airway opposite to the point where there were signs of heating in the return, from the waste monitoring pipes laid over the pack. This model shows why this may occur.

PRESSURE DISTRIBUTION

HIGH PRESSURE



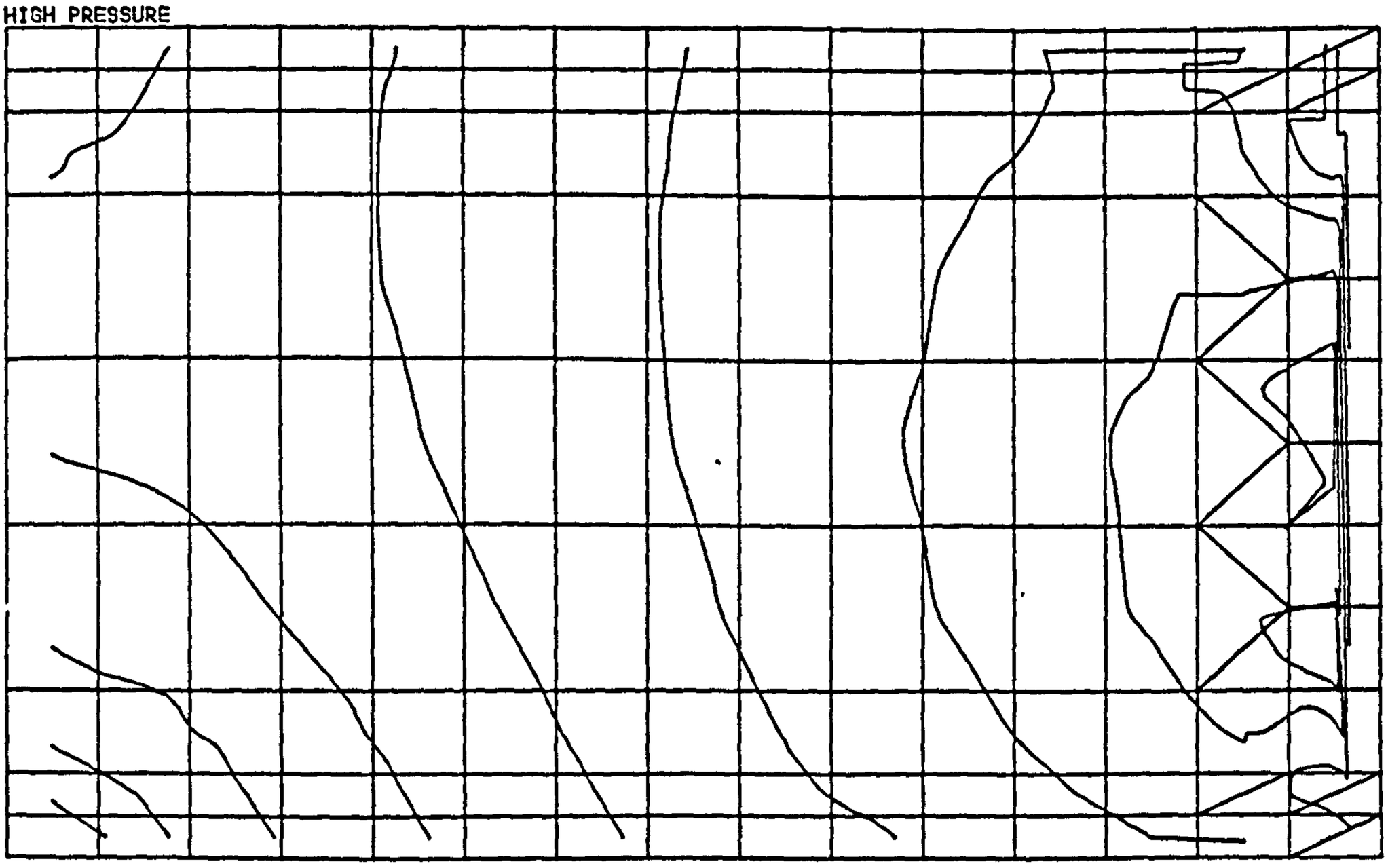
DATA:0 TO 164 CONTOUR INTERVAL=16.4

LOW PRESSURE

Figure 7.8a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:1.72E-4 TO 0.006469 CONTOUR INTERVAL=6.3E-4

LOW PRESSURE

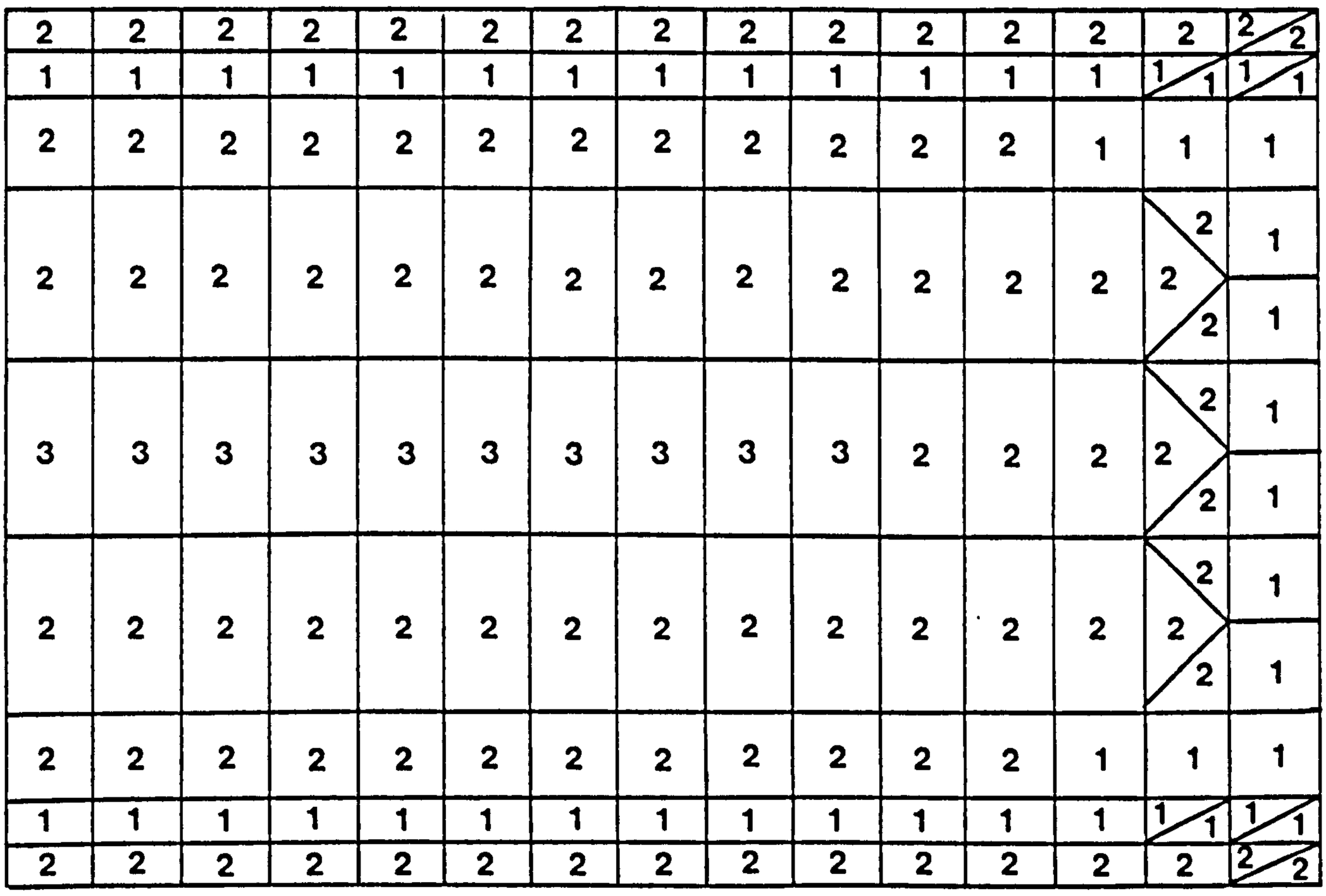
Figure 7.8b Harworth 1's Constant Permeability

However, the unconsolidated conditions that result in high leakage through the waste are found in relatively few mining areas in the UK

7.4.2.1.2 Varying Permeability

The permeabilities in this model were selected so as to give the lowest permeability in the centre of the waste, increasing towards the perimeter. The permeability is symmetrical about the centre of the waste. Four permeability zones, as defined earlier, were used. This results in a large step change in permeability from element to element towards the centre of the mesh. Figure 7.9 shows the element permeabilities.

The pressure distribution is shown in Figure 7.10a. There is a large pressure drop in the middle of the waste, due to the low permeability in this area. Air entering this zone would flow parallel to the face line. Elsewhere the pressure gradient is not so great. The pressure contours are at an acute angle to the face line on either side of the centre of the waste. In the high permeability corridor next to the pack, the contours run almost at right angles to the roadway. This distribution of pressure contours would result in the air flowing diagonally towards the face line from the intake side, and then diagonally from the face line to the return, low pressure side. This is especially pronounced at the face ends. In the corridor next to the pack, the airflow will be parallel to the roadway, with a slight tendency to move into the waste on the intake side, and to leak back into the return side from that corridor.



Permeabilities

1 : $10^{-6}m^2$

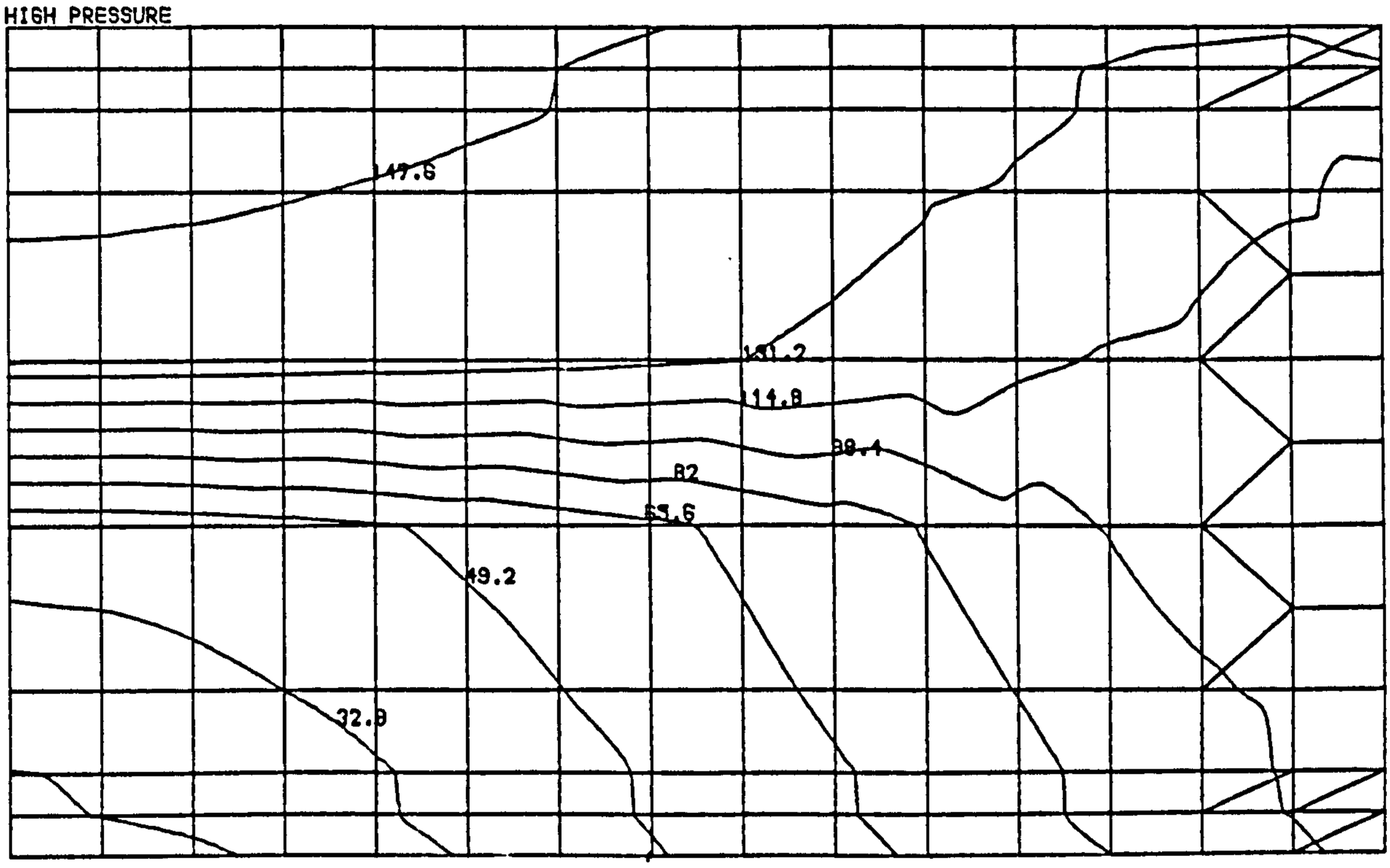
2 : $10^{-7}m^2$

3 : $10^{-8}m^2$

Figure 7.9 Element Permeabilities Used in Varying Permeability Model

PRESSURE DISTRIBUTION

HIGH PRESSURE



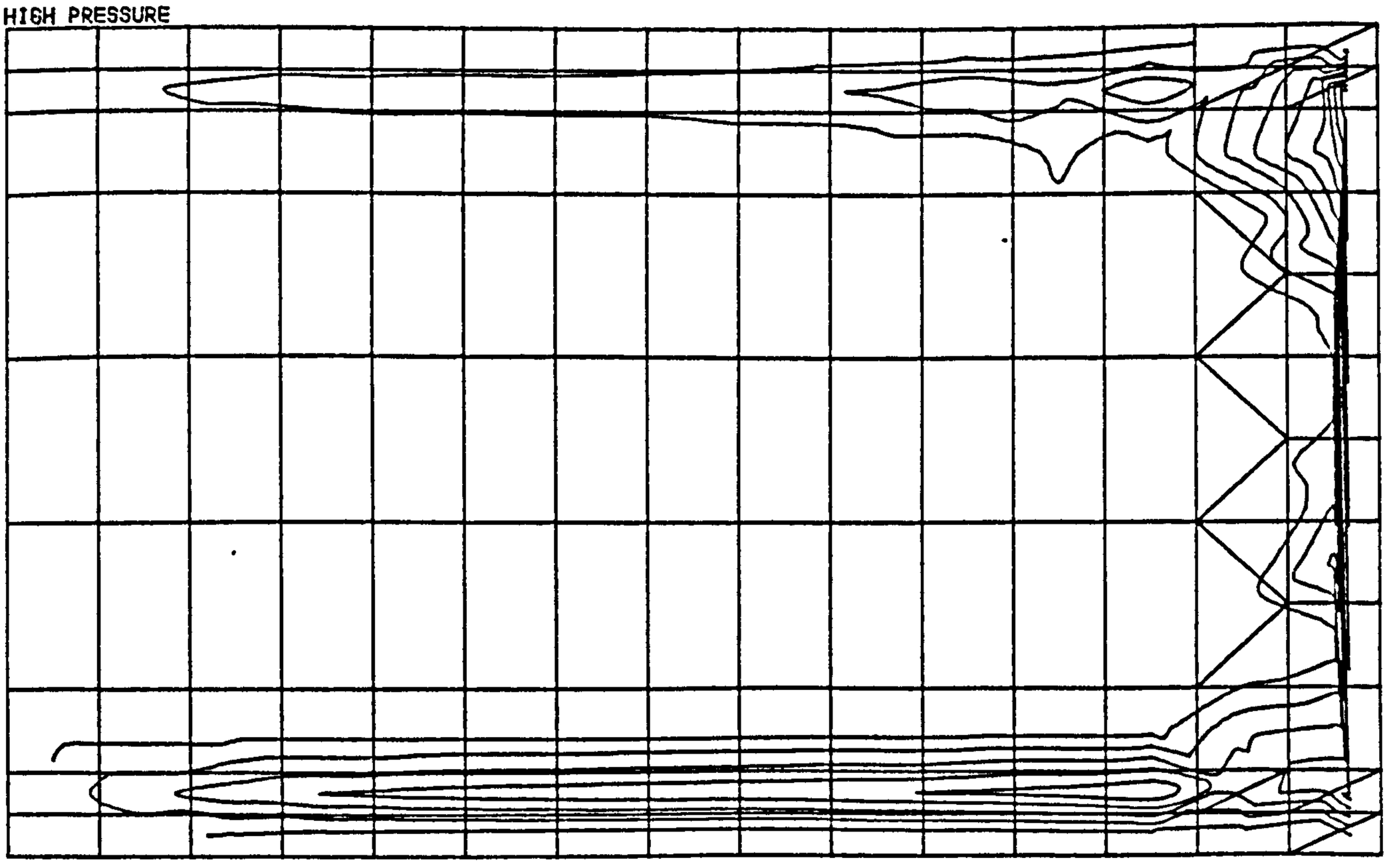
DATA:0 TO 164 CONTOUR INTERVAL=16.4

LOW PRESSURE

Figure 7.10a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:4.58E-4 TO 0.038166 CONTOUR INTERVAL=0.0037708

LOW PRESSURE

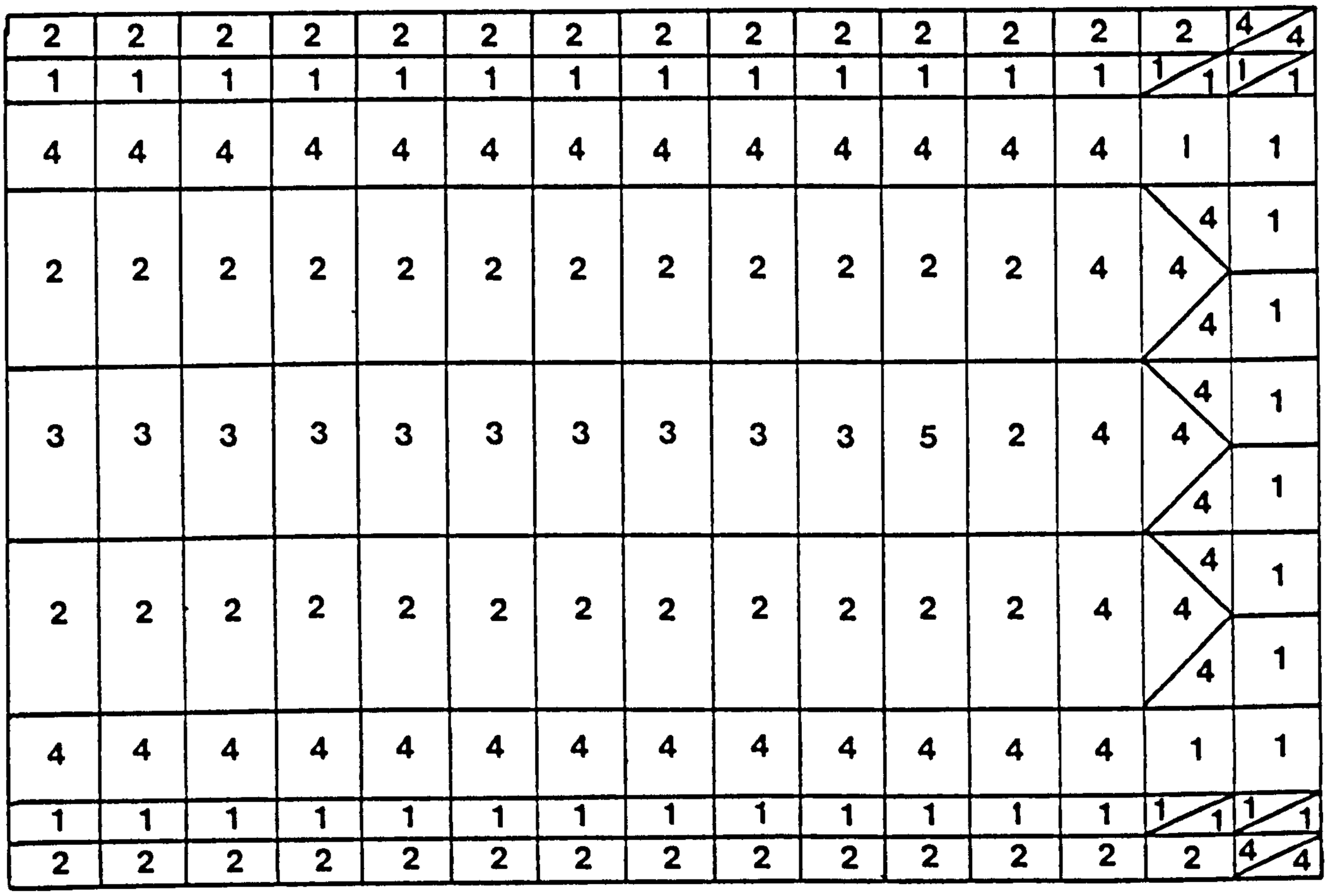
Figure 7.10b Harworth 1's Varying Permeability

The velocity contour plot is in Figure 7.10b. Despite the high pressure gradient in the centre of the waste, there is little flow. Most of the flow occurs in the corridor next to the pack, and immediately behind the face line.

The airflow velocities range from 0.0004m/s to 0.0382m/s. If an average particle diameter of 0.01m is taken, the Reynolds number is from 0.4 to 38.2. Clearly, the flow is not laminar in some zones of the waste. These zones are the pack corridors, and the zone immediately behind the face. While the velocities, in the areas, as calculated, are in error, the model provides a good approximation of the flow in the majority of the waste. The actual velocity in the areas subject to turbulent flow will be somewhat lower than that calculated using the linear, laminar flow, relation.

7.4.2.1.3 Graduated Permeability

The large step change in permeability from element to element in the previous exercise is not realistic. Permeability in the waste changes gradually towards the centre. The exception is the transition of from the permeability of the roadside pack to that of the unconsolidated corridor next to the pack. The permeabilities of the intermediate elements between the corridor and the centre of the waste were altered to give a more gradual rise. The element permeabilities are shown in Figure 7.11.



Permeabilities

1 : $10^{-6}m^2$

2 : $9 \times 10^{-8}m^2$

3 : $10^{-8}m_2$

4 : $3.03 \times 10^{-7}m^2$

5 : $3 \times 10^{-8}m^2$

Figure 7.11 Graduated Permeability Model Element Permeabilities

The pressure distribution, Figure 7.12a, does not differ significantly from that in the previous exercise. Again there is a large pressure drop in the centre of the waste, and flow parallel to the face line. Flow is diagonal from the intake to the face, and from the face to the return, especially around the face ends. In the unconsolidated corridor, flow is virtually parallel to the roadways. The velocity distribution, Figure 7.12b, is slightly different to that in Figure 7.10b. The same trends can be seen, with the majority of flow occuring in the area immediately behind the face line and in the pack corridor. However, higher airflow velocities exist slightly deeper into the waste, and so a greater quantity of air flows deeper in the waste.

The Reynolds number ranges from 0.6 to 37.4.

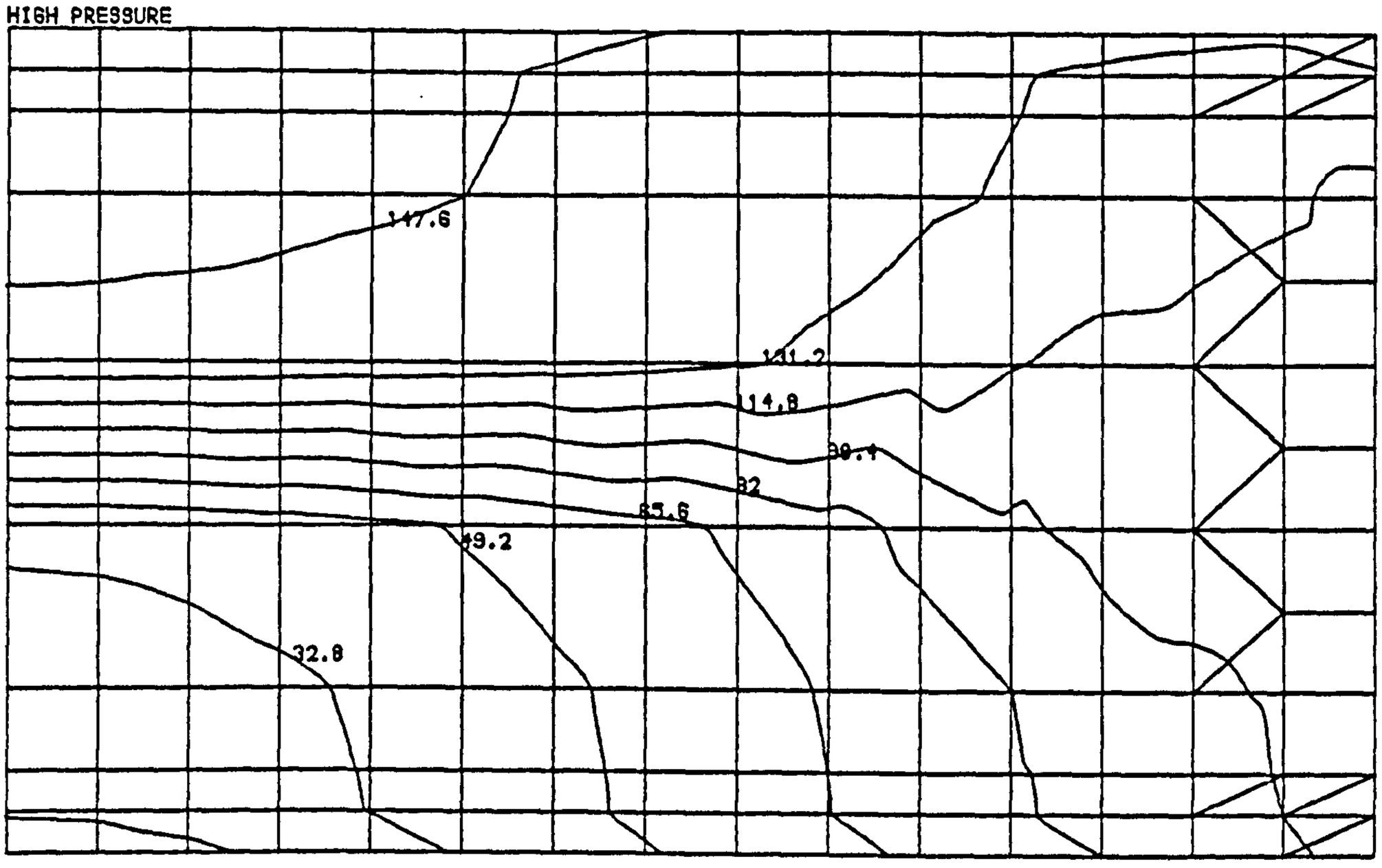
7.4.2.1.4 Faulted Ground

Many incidents of spontaneous combustion are associated with faults and strata discontinuities. This especially applies to those that run from the intake to the face line. Consolidation is reduced around a fault, resulting in increased permeability. Along fault lines there is a large amount of finely divided carbonaceous material, which is able to initiate and sustain a fire.

A fault was modelled by increasing the permeability of the elements along the fault line, with the permeability in the rest of the mesh as in the Graduated Permeability exercise.

PRESSURE DISTRIBUTION

HIGH PRESSURE



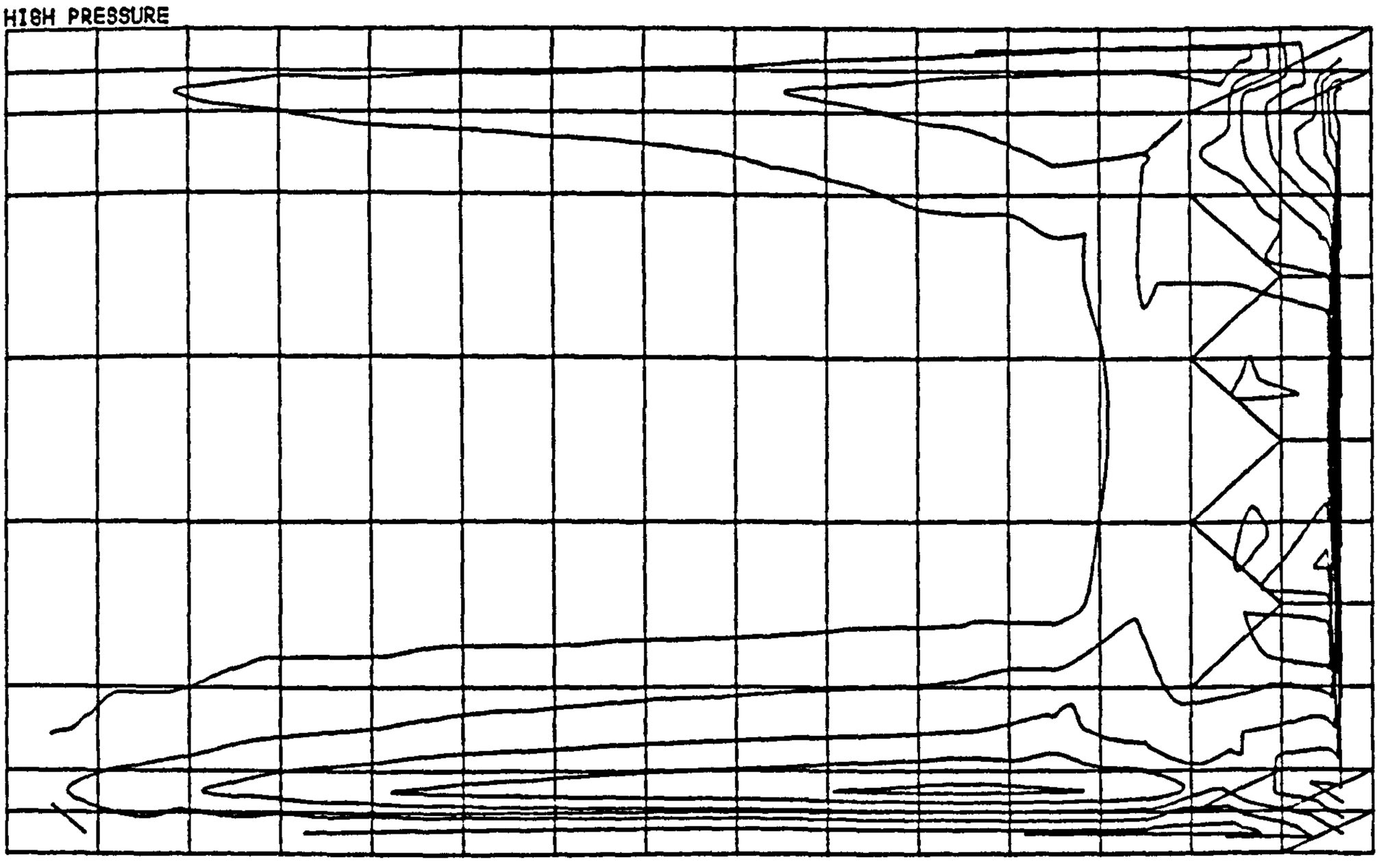
DATA:0 TO 164 CONTOUR INTERVAL=16.4

LOW PRESSURE

Figure 7.12a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:6.E-4 TO 0.038 CONTOUR INTERVAL=0.0037448

LOW PRESSURE

Figure 7.12b Harworth 1's Graduated Permeability

The pressure distribution in shown in Figure 7.13a, with the elements affected by the fault marked by the heavy line. The pressure distribution obtained is no different to that from the Graduated Permeability exercise.

The velocity distribution, Figure 7.13b, is very different. The velocity along the elements affected by the fault is greatly increased. Elsewhere the flow pattern is the same. The Reynolds number ranges from 0.5 to 66.1. The flow around the fault line is not laminar. Having been calculated using laminar flow assumptions, the actual velocity along the fault line may be lower. However, the point is that there is a large amount of airflow through a region of finely broken material in the waste, an obvious spontaneous combustion risk.

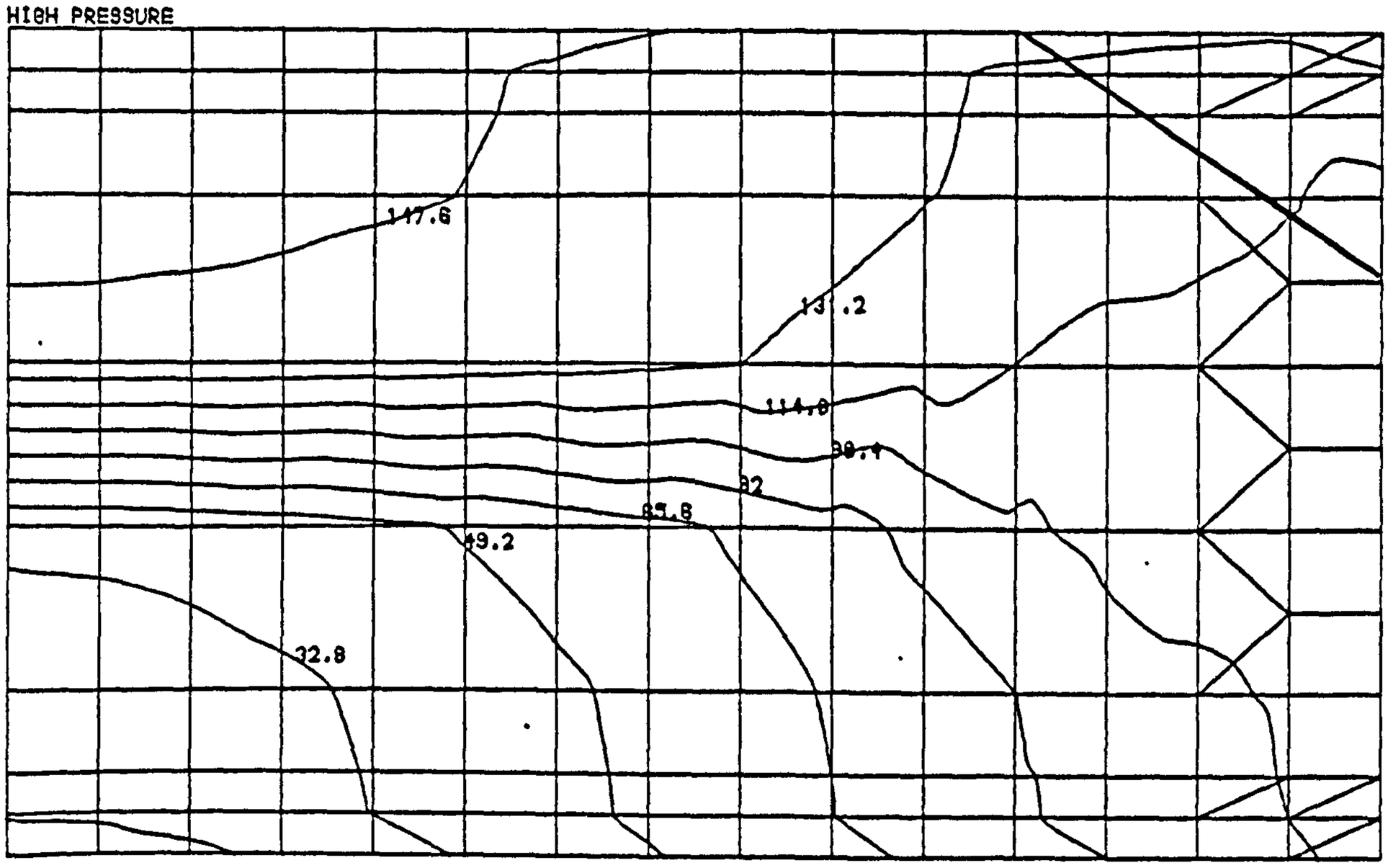
7.4.2.1.5 Nitrogen Injection

Nitrogen use to control spontaneous combustion has been discussed in Chapter 4. The finite element program allows a point source to be applied at a node. This was used to simulate nitrogen injection into the waste. The figures used for nitrogen injection were taken from Figure 7.4, (30) and applied to the boundary pressure and element permeability in the Graduated Permeability exercise.

When nitrogen is injected into the waste, the pipes only extend just over the packs, and not any distance into the waste.

PRESSURE DISTRIBUTION

HIGH PRESSURE



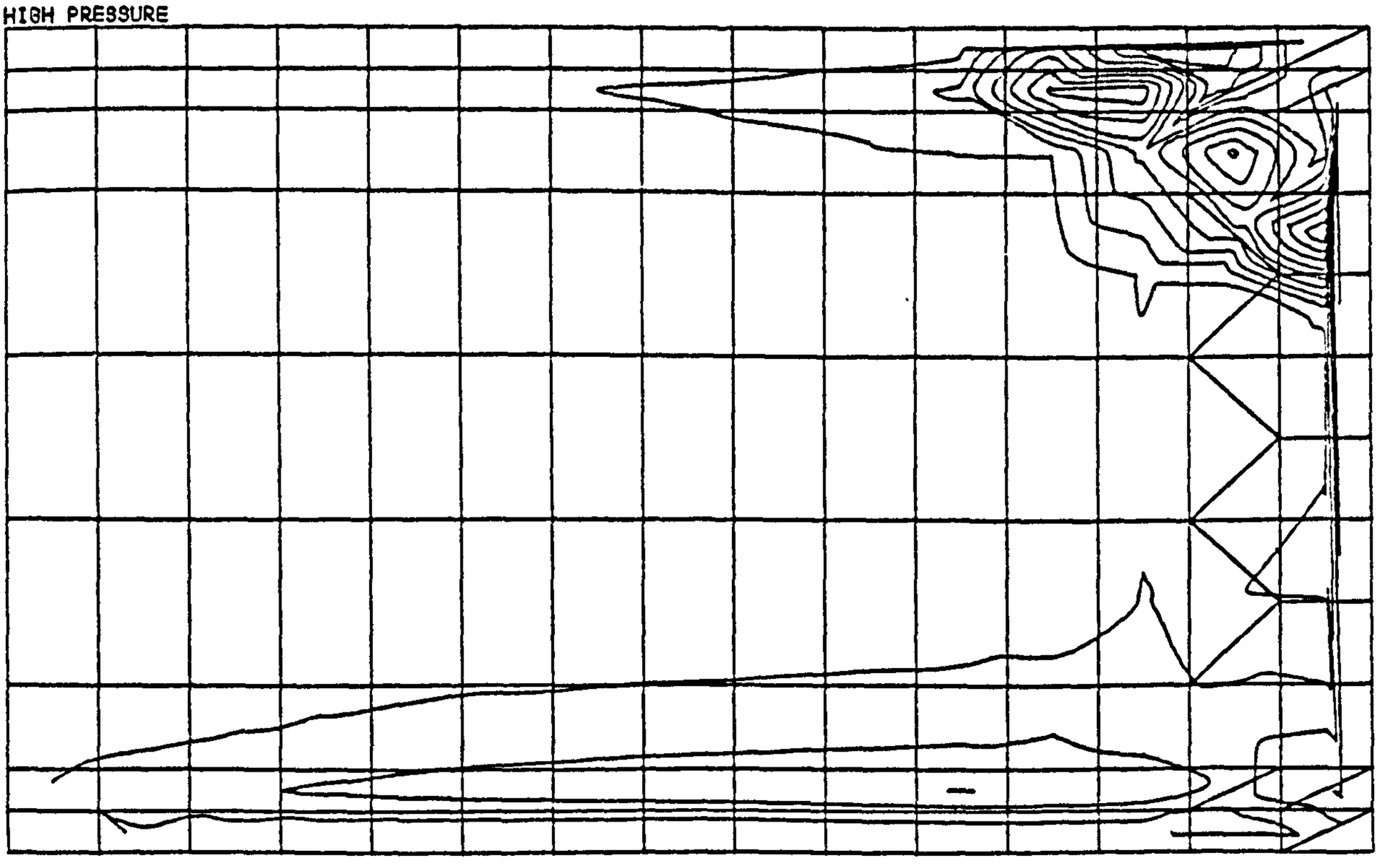
DATA:0 TO 164 CONTOUR INTERVAL=16.4

LOW PRESSURE

Figure 7.13a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:5.88E-4 TO 0.066173 CONTOUR INTERVAL=0.0065585

LOW PRESSURE

Figure 7.13b Harworth 1's Intake Fault

The nodes selected were those immediately on the waste side of the pack.

The resulting pressure distribution is shown in Figure 7.14a. Injecting nitrogen into the waste raises the local pressure around the injection site, to such a degree that the normal pressure distribution is completely disrupted. The pressure at the injection site is higher than that in the roadway. It is immediately apparent that gas will flow back out of the waste , and into the roadway. This serves to explain and confirm the observations that were made at Daw Mill, Chapter 5, where nitrogen reemerged into the roadway around the injection hole.

From the two injection points furthest from the face, some gas flows back into the intake around the injection site. Of the rest of the gas, a fraction migrates across the waste to the return, some flows up the unconsolidated corridor, away from the face, while the remainder migrates diagonally toward the return face end. The two injection sites that are nearest to the face have much higher flow rates, resulting in a greater pressure gradient. The majority of the gas flow is in two directions. There is leakage back into the intake roadway, while the balance passes through the waste to emerge at the intake side of the face. A small amount migrates across the waste, in the normal airflow pattern.

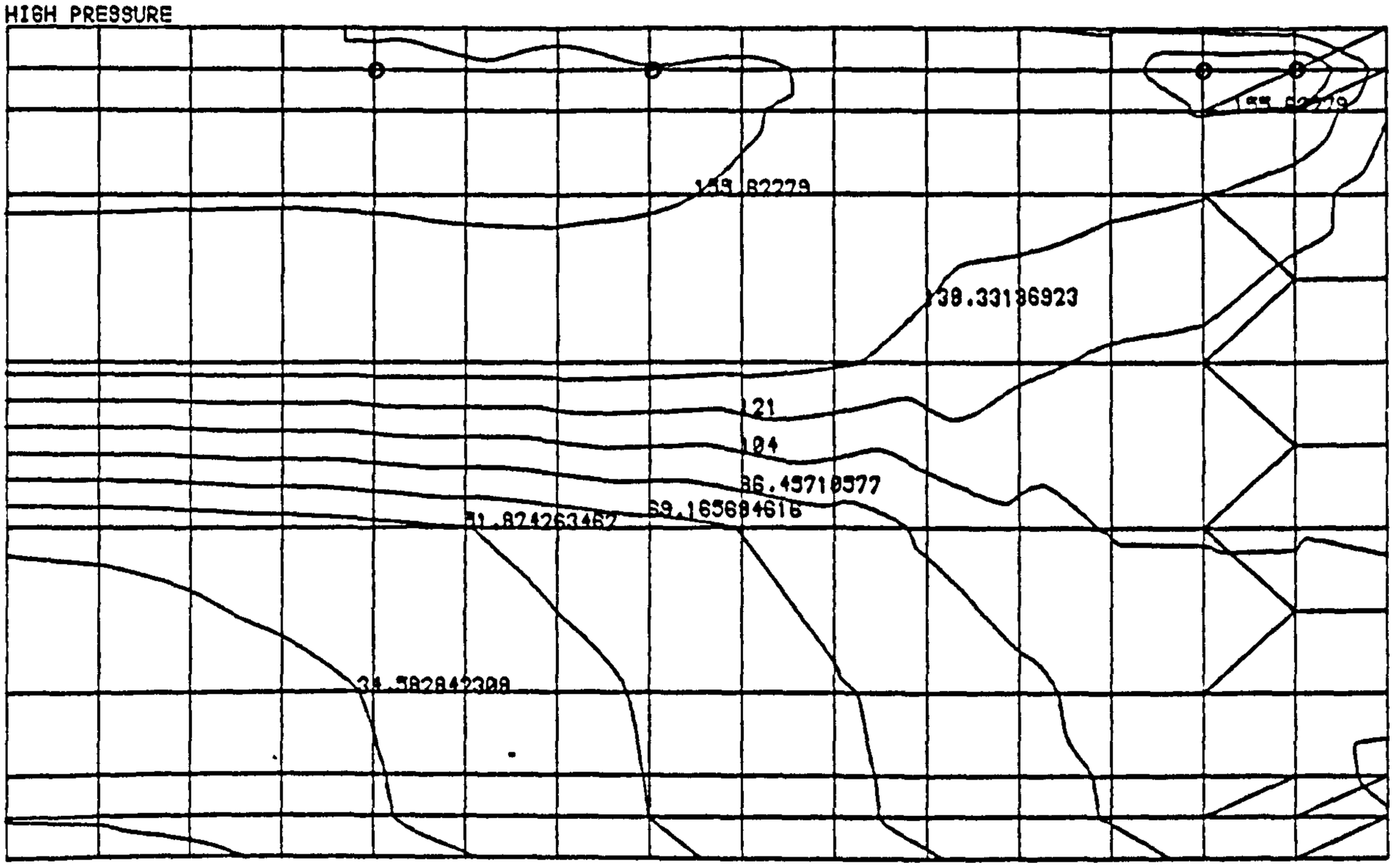
The flow pattern on the return side of the waste is only slightly distorted.

The velocity contour plot, Figure 7.14b, shows the high velocities around the injection sites. Taken in conjunction with the pressure distribution it reveals how little injected nitrogen flows deep in the waste, and the local nature of the majority of the flow. The Reynolds number ranges from 0.5 to 158, so the nitrogen flow is turbulent.

The flow pattern, and velocities are very different from Figure 7.4. As no details have been published on this simulation it is not possible to comment on the source of the differences. It is apparent, though, that the nitrogen injection has not been considered as a point source raising the local pressure. If higher permeabilities are used, the pressure is not raised as much around the injection point, and the pressure distribution will more closely resemble that obtained in the Graduated Permeability exercise. Imposing nitrogen flow onto this model, and assuming little disruption of the pressure distribution will still not yield the flow pattern in Figure 7.4.

PRESSURE DISTRIBUTION

HIGH PRESSURE



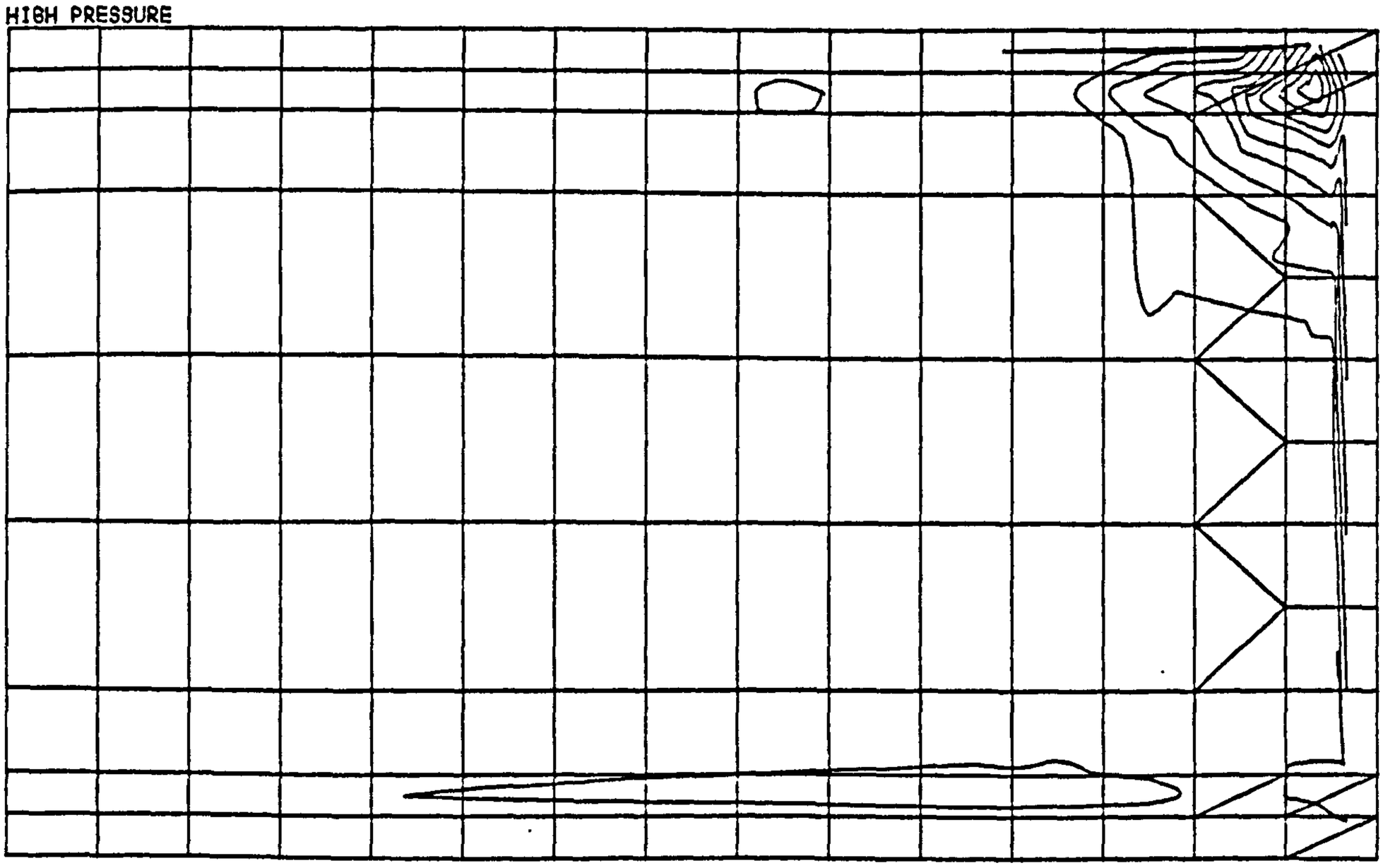
DATA:0 TO 172.91421154 CONTOUR INTERVAL=17.291421154

LOW PRESSURE

Figure 7.14a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:1.89E-4 TO 0.157 CONTOUR INTERVAL=0.0156842

LOW PRESSURE

Figure 7.14b Harworth 1's Nitrogen Injection

7.4.2.2 Harworth 3's

Harworth 3's is an advance face. The face had advanced about 1500m from the face start line, the face line was 250m long, and the height of extraction was 1.8m. Coal was conveyed in the return. A slightly larger area has been modelled, 400m of gate roadway leading to the face, and the 250m long face. There were no major faults that crossed the airways. As air quantity measurements were taken it was possible to compare the actual increase in air quantity in the return, with that obtained from the velocity values calculated in the finite element program.

The exercises carried out were the Graduated Permeability model, the fault on the intake side, and nitrogen injection.

7.4.2.2.1 Graduated Permeability

A very similar pattern to that on Harworth 1's (Figure 7.12a) is obtained. The pressure drop around the district is 70% greater, and the proportion of the district pressure drop on the face is larger. This accounts for the slightly different distribution at the waste perimeter. The pressure drop along the face line was 125 Pa, of which 68 Pa was lost in the 75m around the shearer. At the time the pressure readings were taken, the shearer had almost completed a cut towards the return end, and was about 30m from the face end. The pressure loss associated with the shearer is a transient that moves with the shearer. As such, it is not valid to redistribute this loss along the face. It is clear that the

position of the shearer has an effect on the leakage, especially around the face ends. In the event of a spontaneous heating on the intake side it would seem a sensible move to position the shearer as far towards the return as possible.

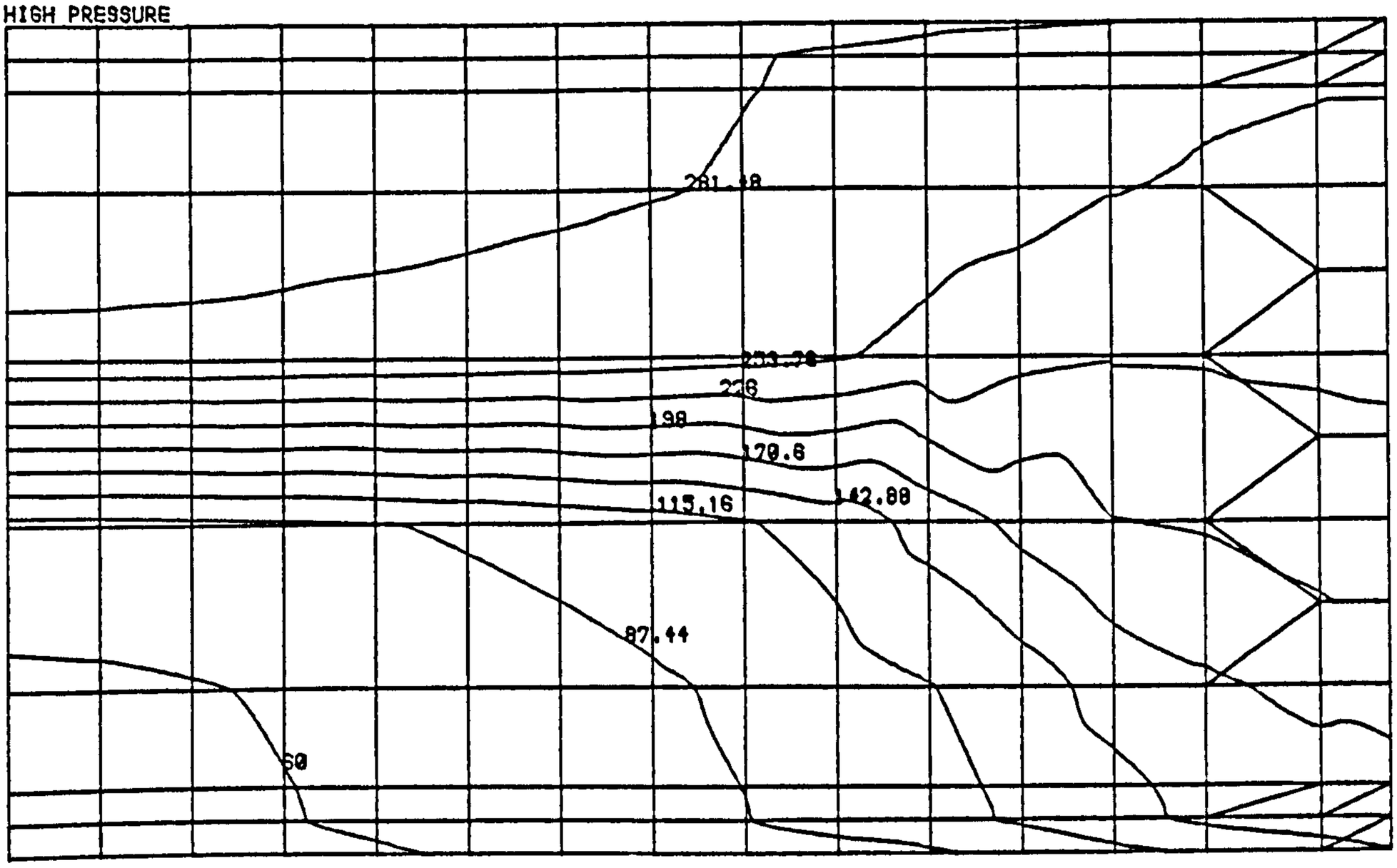
The pressure distribution is shown in Figure 7.15a. The flow paths are even more pronounced than earlier. Air moves across the corners of the waste at the face ends, parallel to the roadways in the unconsolidated pack corridor, and parallel to the face line behind the face.

The velocity distribution, Figure 7.15b, is very similar to that in Figure 7.12b. The velocity at the face end is higher due to the high pressure loss around the shearer. In other parts of the area the pressure gradient tends to be higher, so the velocity is raised. The Reynolds number ranges from 1.2 to 50.

It is possible to include a different, non linear, flow regime in a finite element program. This would then calculate pressure differences and flows both in the area where laminar flow occurs, and the areas of non laminar flow. This is done using an iterative method where the non linear relationships are modelled as piecewise linear approximations.

PRESSURE DISTRIBUTION

HIGH PRESSURE



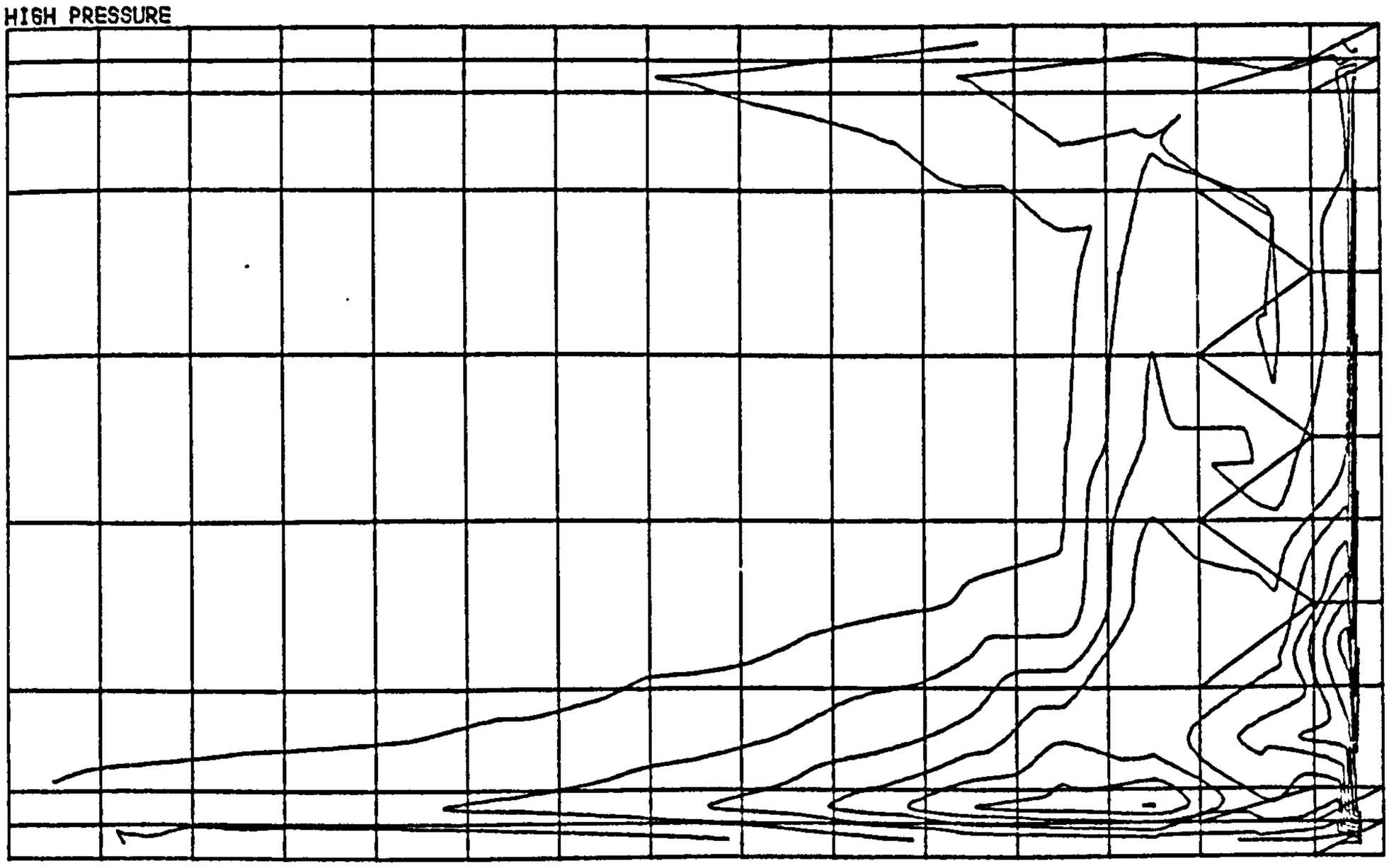
DATA:32 TO 309.2 CONTOUR INTERVAL=27.72

LOW PRESSURE

Figure 7.15a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:0.0012 TO 0.05 CONTOUR INTERVAL=0.0048538

LOW PRESSURE

Figure 7.15b Harworth 3's Graduated Permeability

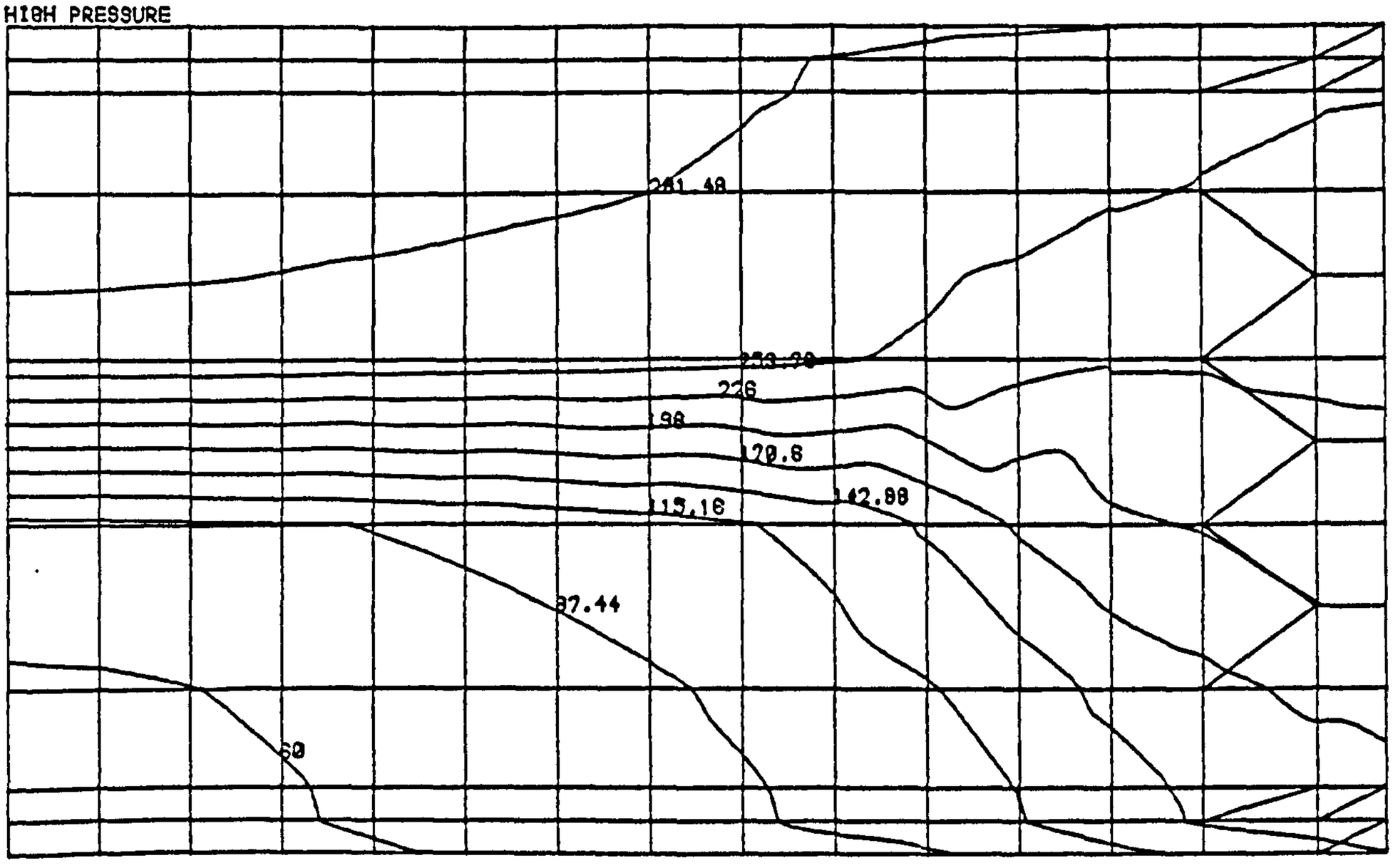
The method requires a considerable increase in the sophistication of the program, and the computer time used. It is the opinion of the author that non laminar flow may occur in limited areas, and so this increase in sophistication is not warranted at this stage.

The measured difference in air quantity in the return between the rip and a point approximately 400m outbye was $1.1m^3/s$. The quantity increase predicted was calculated by taking the average Y velocity along an element side, and multiplying this by the estimated area of flow into the roadway along that element side. The area is the length of the element multiplied by 1.5m, an estimate of the consolidated height of the pack.

The quantity increase predicted was 3.7 m^3/s. This showed that the permeabilities used were of the right order of magnitude, but the wrong value. The velocities could be reduced to one third of the calculated values by reducing the permeabilities throughout the mesh. This was done, and the results are shown in Figure 7.16a and 7.16b. The pressure distribution, and hence the airflow pattern were not changed, but the velocity values are reduced. The Reynolds number now ranged from 0. 5 to 17, so the flow is still non laminar in some areas. The following exercises were all conducted using this reduced permeability.

PRESSURE DISTRIBUTION

HIGH PRESSURE



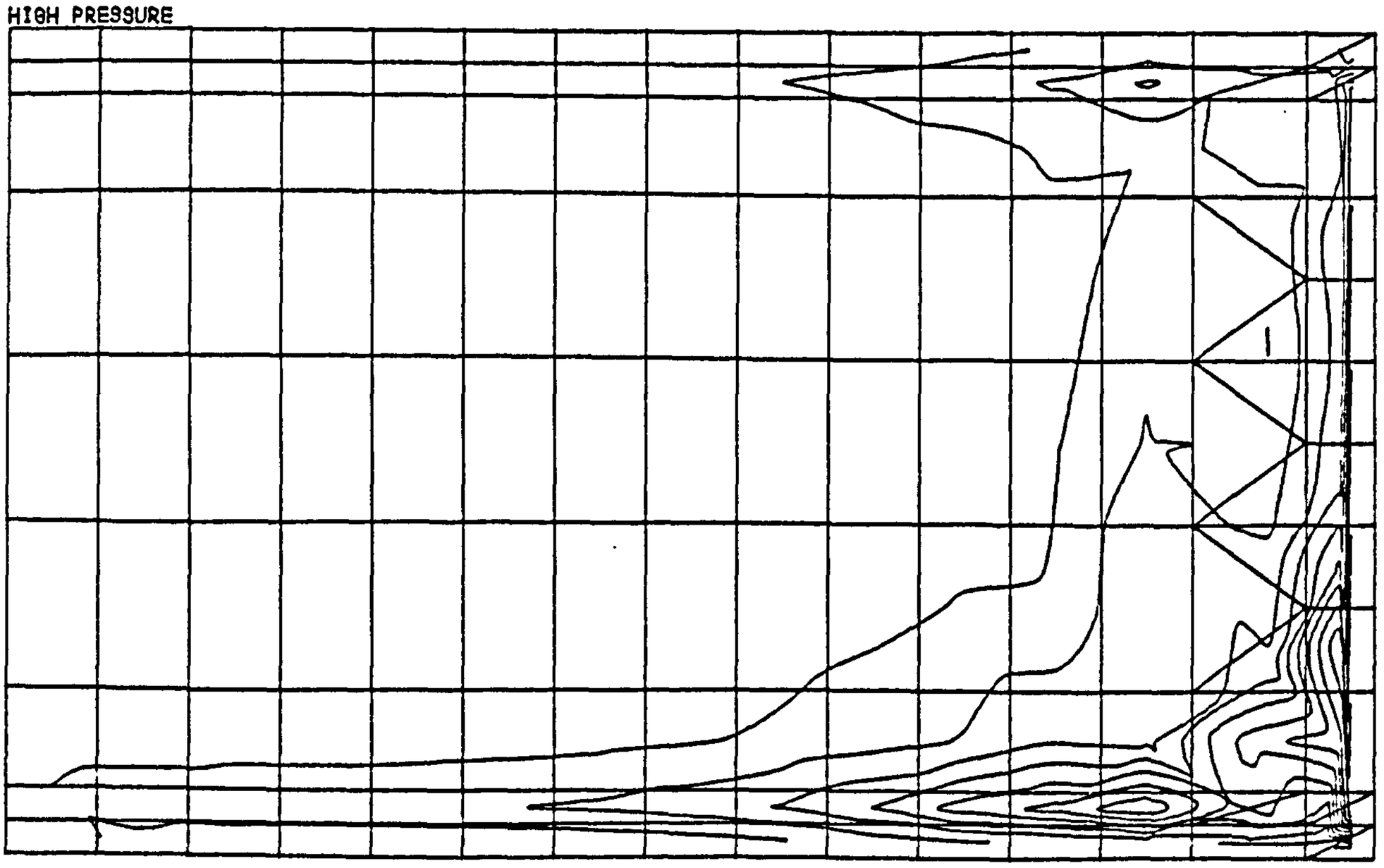
DATA:32 TO 309.2 CONTOUR INTERVAL=27.72

LOW PRESSURE

Figure 7.16a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:5.14E-4 TO 0.016738 CONTOUR INTERVAL=0.0016224

LOW PRESSURE

Figure 7.16b Harworth 3's Revised Permeability

7.4.2.2.2 Intake Fault

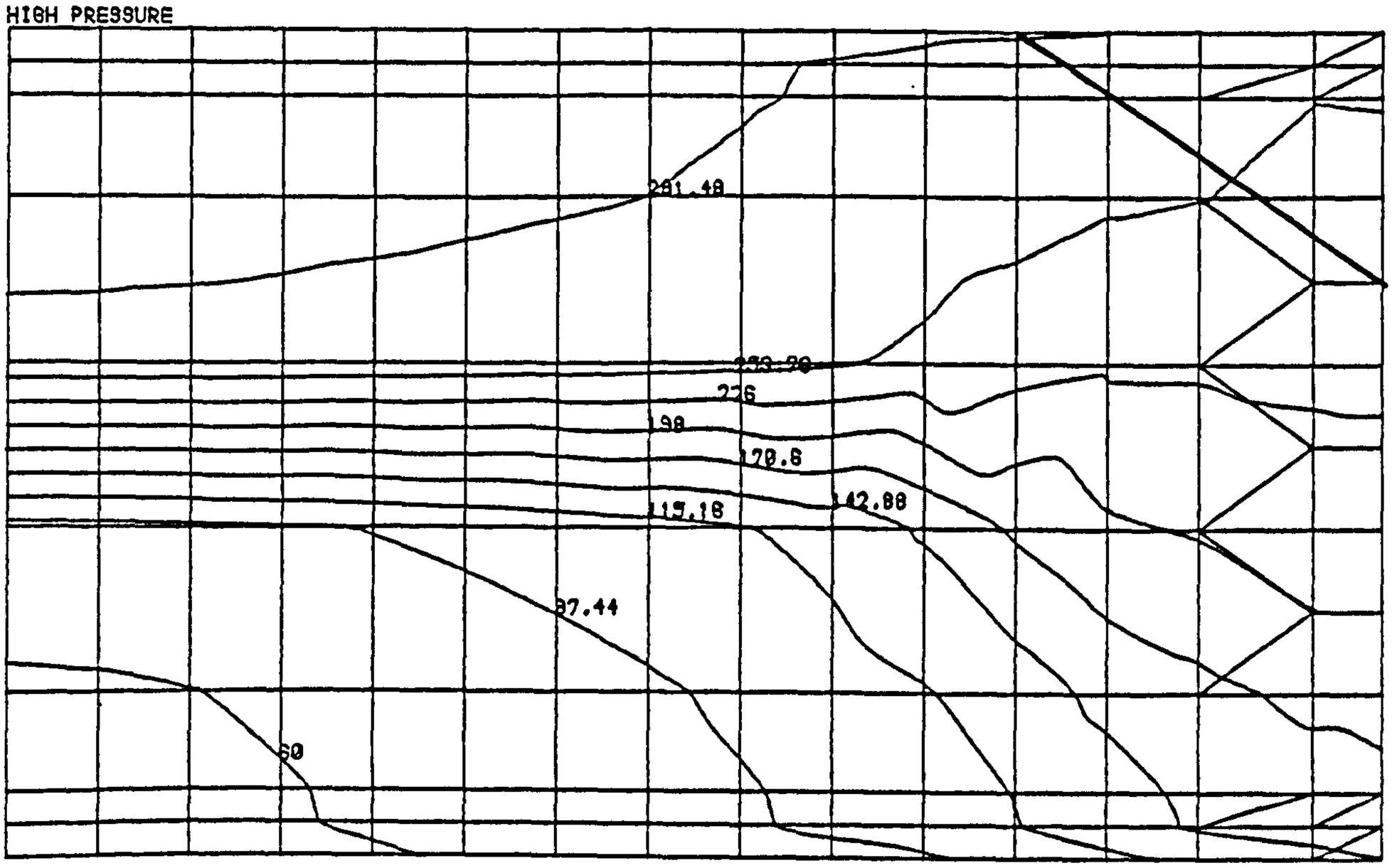
The pressure contour plot is shown in Figure 7.17a, and the velocity distribution in 7.17b. These results are very similar to those obtained using the boundary pressures from Harworth 1's. The pressure difference across the intake face end was higher, but because of the reduced permeability the velocity along the fault line was lower. The Reynolds number was from 0.5 to 28

7.4.2.2.3 Nitrogen Injection

The same nitrogen injection quantities were used as in the earlier model. The same pattern can be observed in Figures 7.18a and 7.18b as in 7.14. The reduced permeability resulted in a greater pressure rise at the nitrogen injection site. This effect was counteracted by the higher boundary pressures, however the injection sites nearest the face have a high enough difference to show how the gas will flow in the waste. The flow is essentially radial from the injection point. The lower pressure difference means that a lower proportion of the nitrogen injected is forced back into the airway immediately around the injection site. A portion travels down the unconsolidated corridor to emerge on the face, some moves away from the face in the corridor, and then across the waste in the normal airflow paths. The velocity contour plot shows the extent to which flow is concentrated around the face end, and how little of the gas flows through the waste. The Reynolds number ranged from 0.15 to 179.

PRESSURE DISTRIBUTION

HIGH PRESSURE



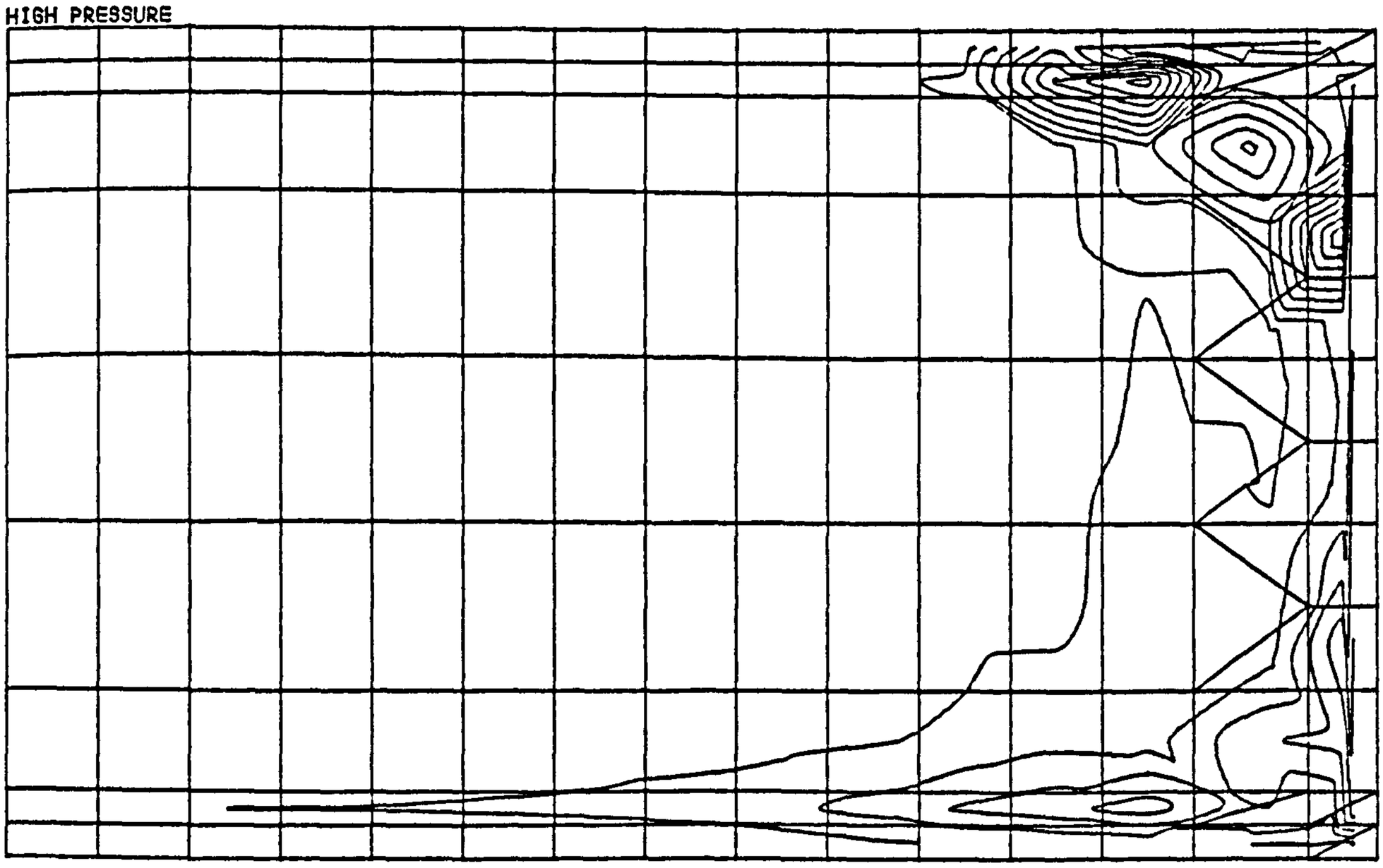
DATA:32 TO 309.2 CONTOUR INTERVAL=27.72

LOW PRESSURE

Figure 7.17a

VELOCITY CONTOURS

HIGH PRESSURE



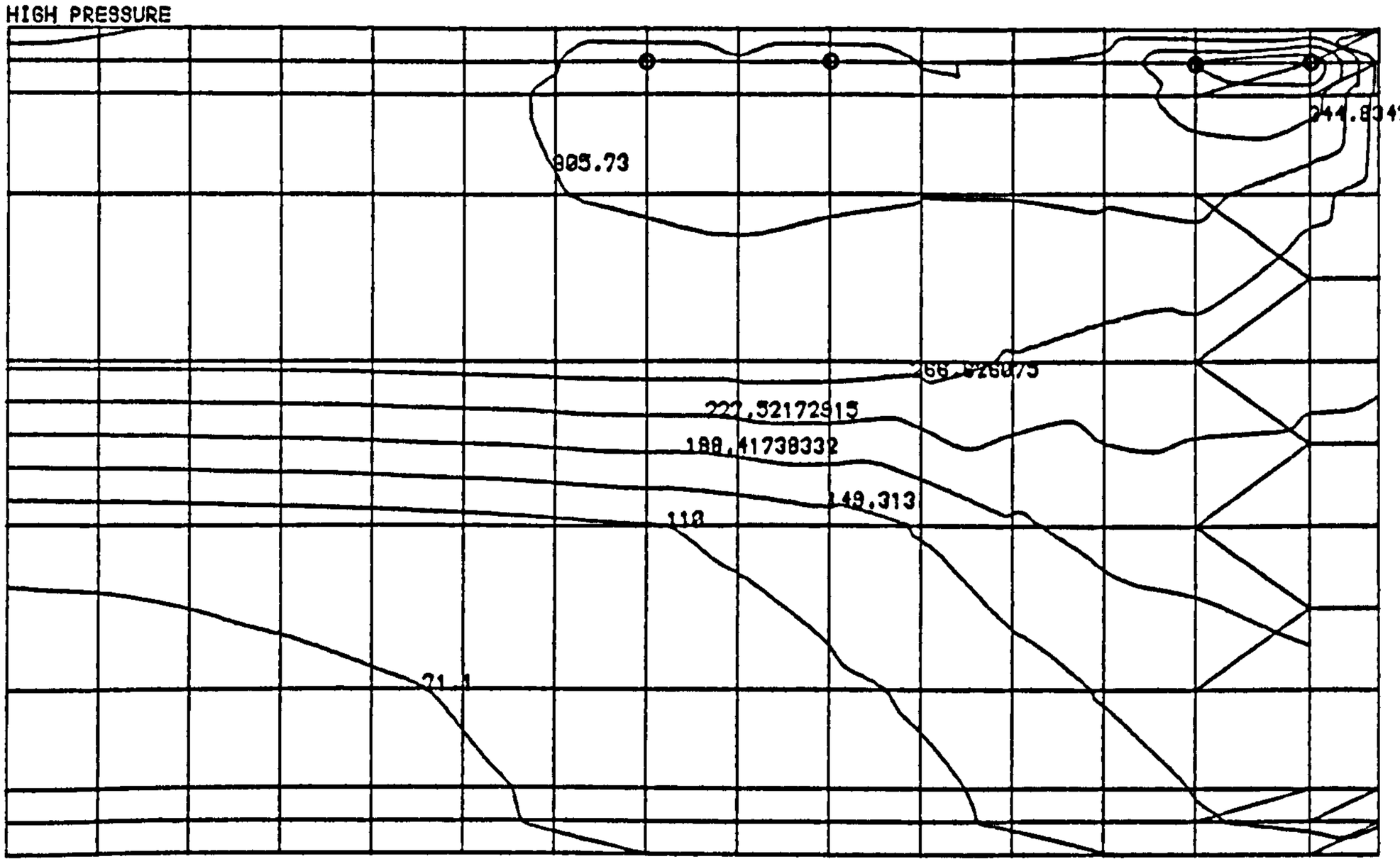
DATA:5.13E-4 TO 0.028542 CONTOUR INTERVAL=0.0028

LOW PRESSURE

Figure 7.17b Harworth 3's Intake Fault

PRESSURE DISTRIBUTION

HIGH PRESSURE



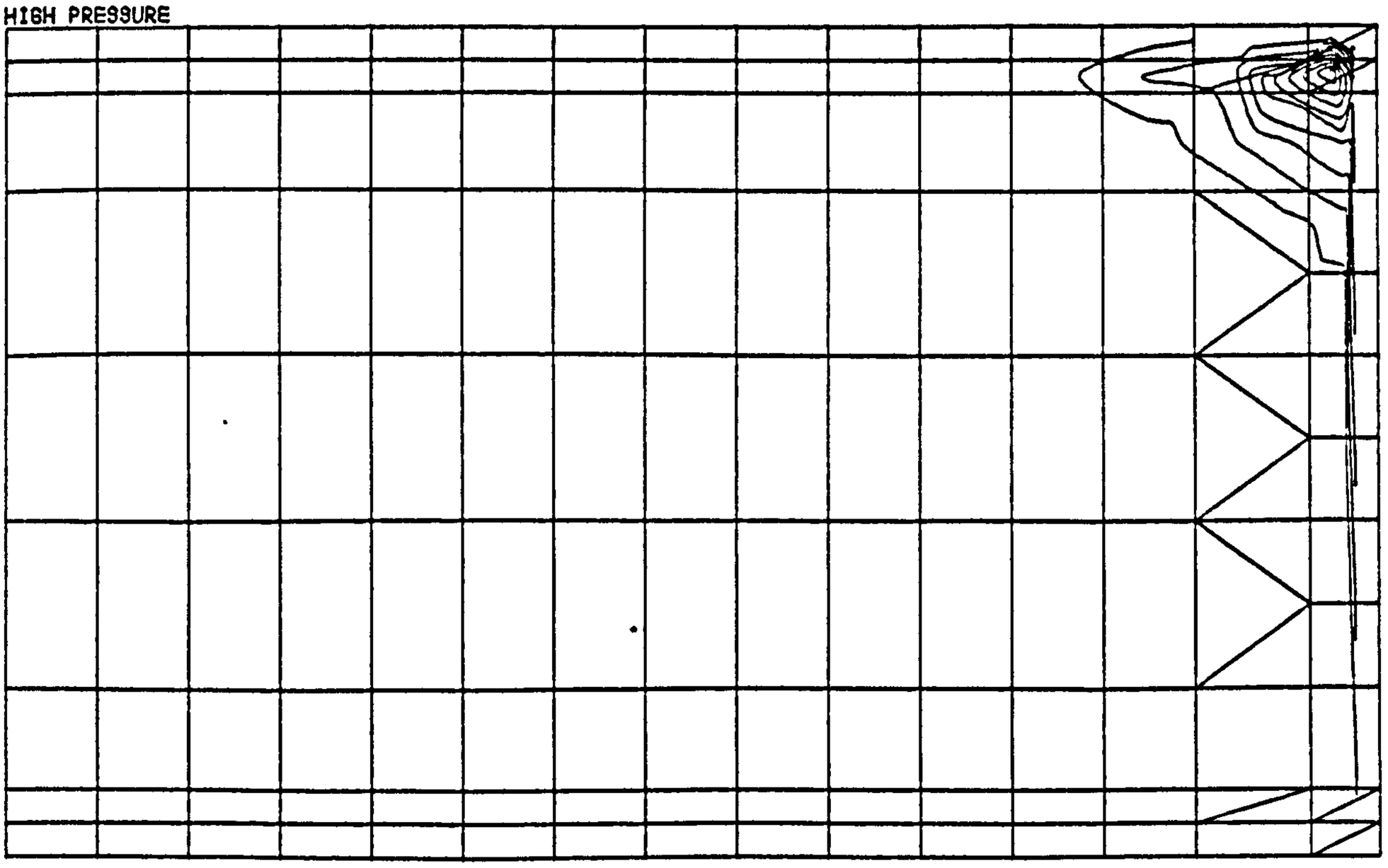
DATA:32 TO 423 CONTOUR INTERVAL=39.1

LOW PRESSURE

Figure 7.18a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:1.47E-4 TO 0.179448 CONTOUR INTERVAL=0.01793

LOW PRESSURE

Figure 7.18b Harworth 3's Nitrogen Injection

7.4.2.3 Retreat Face

No pressure readings were taken on a retreat face. It was felt, though, that it would be a useful exercise to examine the pressure distribution in the waste of a retreat face. The same mesh was used, but the permeabilities in the mesh were changed. The old roadways have a high permeability, as they do not close and consolidate fully because of the support provided by the solid ribside. Permeability decreases towards the centre of the waste. Along the faceline, the permeability will be similar to that on an advancing face.

Boundary pressures used along the face line were the same as those from Harworth 3's.

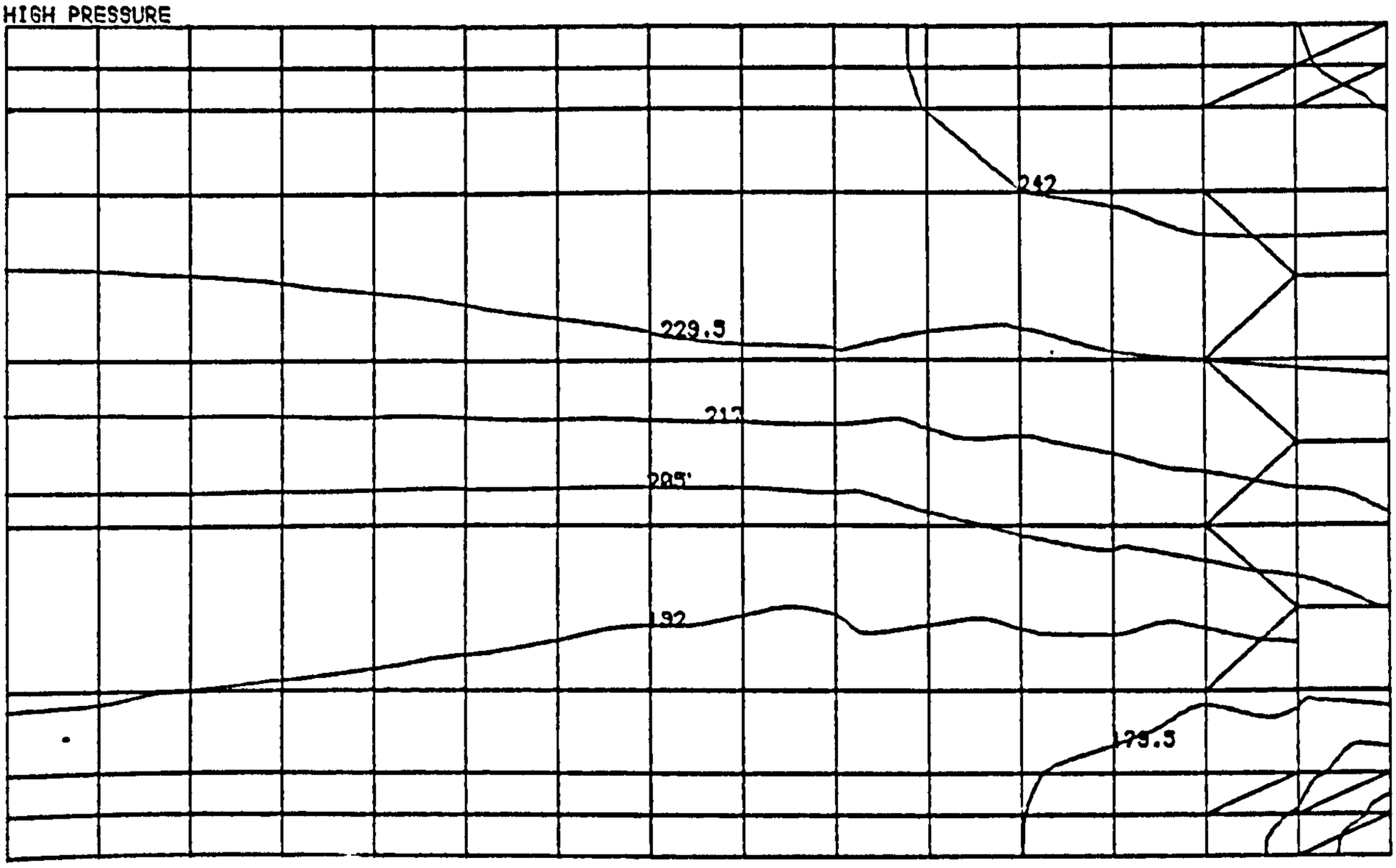
The resulting pressure and velocity distributions are shown in Figures 7.19a and 7.19b. The Russian workers calculated the equations of the streamlines in the waste of a retreat face. The theoretical calculations gave streamlines that were parabolas, with most of the flow occuring immediately behind the face. (56)

Examination of the pressure contours shows that the air enters the intake on the intake side, and follows a curved path back toward the return. The position of the shearer has a great effect on the pressure distribution.

The velocity contour plot gives most of the flow in the region behind the face, as would be expected. The Reynolds number ranges from 0.20 to 76

PRESSURE DISTRIBUTION

HIGH PRESSURE



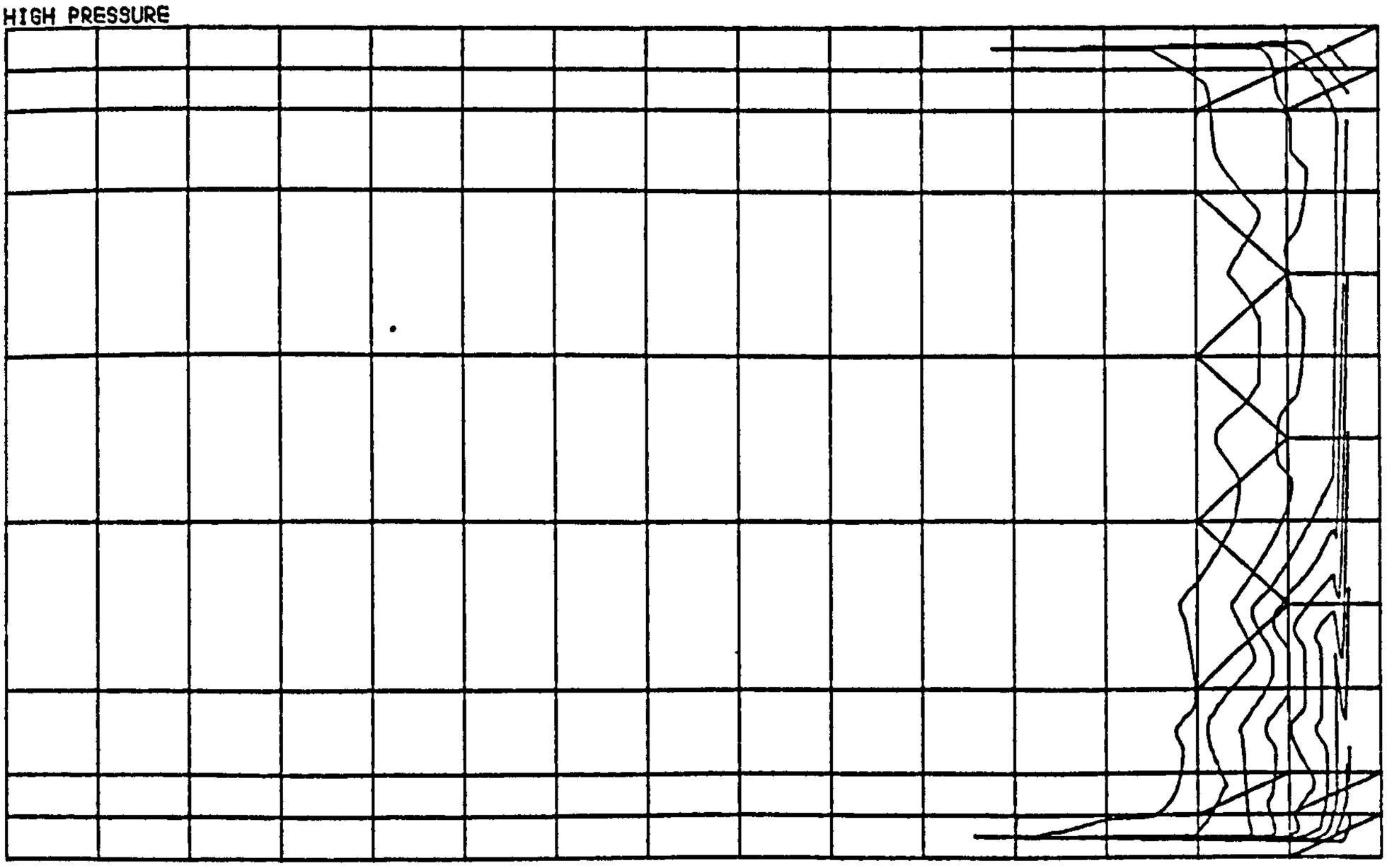
DATA:142 TO 267 CONTOUR INTERVAL=12.5

LOW PRESSURE

Figure 7.19a

VELOCITY CONTOURS

HIGH PRESSURE



DATA:2.42E-4 TO 0.076 CONTOUR INTERVAL=0.0075738

LOW PRESSURE

Figure 7.19b Retreat Face

7.4.3 AREA AT RISK ON AN ADVANCE FACE

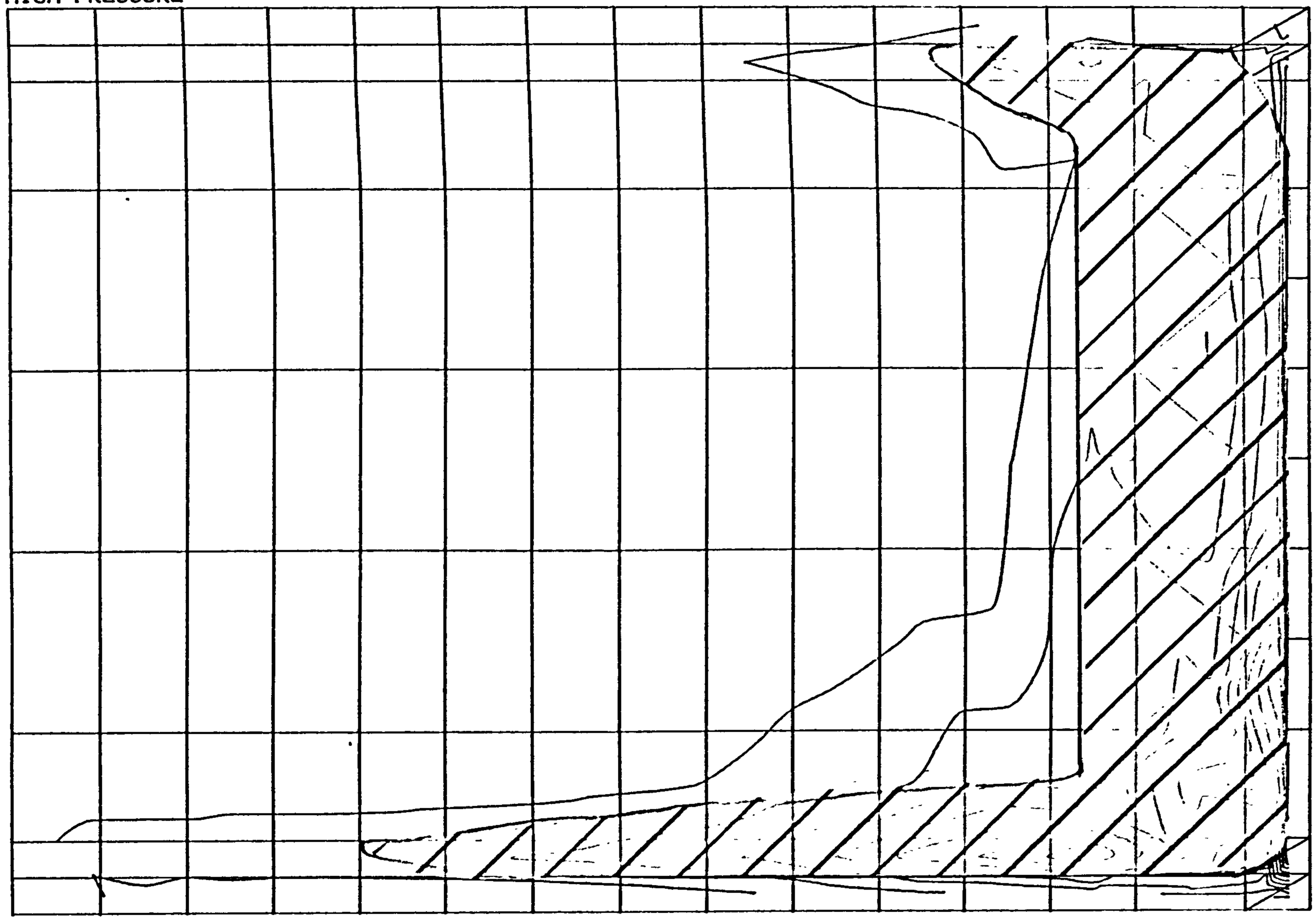
Evseev has proposed that to initiate spontaneous combustion an airflow speed of between 0.1 and 0.9 m/min, and an oxygen content of greater than 17% are required. (52)

These figures can be combined with the results from Hucknall (Chapter 6), and the velocity contour plot from the Graduated Permeability model on Three's, using the reduced permeabilities, to postulate an area at risk on an advance face. This area is shown on Figure 7.20 as the shaded box.

The velocities in the unconsolidated corridor and immediately around the face ends are too high for these criteria. At a depth of greater than 70m behind the face line, extapolating the results from Hucknall gives an oxygen content of less than 17%

As mentioned earlier, the flow in the unconsolidated corridor and at the face ends, may be turbulent. If this is the case, then the velocities calculated assuming laminar flow will be too high, and the area at risk may extend into these zones.

HIGH PRESSURE



DATA:5.14E-4 TO 0.016738 CONTOUR INTERVAL=0.0016224

LOW PRESSURE

Figure 7.20 Area at Risk of Spontaneous Combustion

7.5 CONCLUSIONS

The theories behind fluid flow through a permeable medium have been developed from the equations governing flow through capillary tubes.

A review has been conducted of past work on the flow of air through the waste. Most of the methods used previously placed constraints on the flow paths of air.

The finite element method has been used to examine the pressure distribution and airflow pattern under differing conditions. The effect of assuming a uniform permeability in the waste is to produce a regular pressure distribution, and velocities that are highest where the pressure gradient is greatest. When a distribution in which the permeability decreases towards the centre of the waste is used, the pressure distribution indicated that that air flowed down the unconsolidated corridors next to the packs, and across the corners of the waste at the face ends. The velocity distribution showed that most of the flow occurs at the perimeter of the waste.

Two special conditions were examined. In the first a line of low permeability to simulate a fault was introduced. This had little effect on the pressure distribution, but increased the air velocity, and hence quantity, alone this line. In the second, point sources of flow to simulate nitrogen injection were introduced. These raised the local pressure around the source, resulting in radial flow away from the source.

A second set of exercises was carried out using a different set of boundary pressures. In these exercises the permeabilities used were reduced to correlate the predicted air leakage into the return with that measured.

An area at risk to spontaneous combustion on an advance face was postulated

The airflow speeds in some areas of the waste were identified as being in the non laminar flow regime.

CHAPTER 8

CONCLUSIONS AND SUGGESTIONS FOR FURTHER WORK

8.1 INTRODUCTION

The overall objectives of the study were to investigate the airflow pattern that might be found in a longwall coalface waste, and thus improve the use of prevention and combat measures against spontaneous combustion, especially that of nitrogen injection.

The project was carried out in three phases. Firstly, a face that was being salvaged and had nitrogen injection continuing as a precaution was found. The flow of nitrogen from the injection hole was identified by the addition of a tracer gas. Gas samples were taken from various points around the district. Secondly, an attempt was made to develop a reliable method of obtaining gas samples from deep in the waste, and to examine the airflow and oxidation and oxidation zones in the waste. This yielded useful data on the composition of the waste gas a short distance behind the face. Finally, pressure distribution in the waste was modelled using the finite element technique.

8.2 BEHAVIOUR OF NITROGEN INJECTED INTO THE WASTE

From the first gas samples taken, high levels of SF_6 were detected. At first it was thought that this was the result of a leak in the introduction apparatus, but subsequent tests showed that this was not the case. SF_6 levels were raised a short distance outbye of the injection site. This suggests that the SF_6, and so the nitrogen, was leaking back into the intake general body air from the injection point in the waste. Samples taken along the intake roadway towards the face showed a fall in SF_6, oxygen and carbon dioxide levels around the site of each injection pipe.

Comparison of the atmospheric pressure and level of SF_6 revealed that the SF_6 level was dependent on the rise and fall of the barometer. As the barometer rose, the SF_6 in the general body samples fell, and as the barometer fell, the level of SF_6 rose. A minimum of 25% and a maximum of 60% of the injected SF_6 was present in the general body samples at the intake rip. This rise and fall of the SF_6 level with the barometer was displayed on all the samples taken along the face line, and at the return rip. The carbon monoxide level followed a similar trend. This shows the effect that the changing barometric pressure has on the composition of the waste gases.

Gas levels in samples from behind the face line rose and fell according to the position from which they were taken. High levels of carbon monoxide were detected at the same place during

nitrogen injection as they were when the face had to be abandoned six weeks later. The change in gas levels along the face line demonstrated the existence of leakage paths through the waste to the face line. This rise of SF_6 along the face proved that some nitrogen from the injection hole monitored was passing through the waste.

8.3 WASTE GAS SAMPLING

Gas Samples were taken from the waste at depths of up to 15m behind the face line. The method used proved unequal to the load imposed in the waste. In order to reliably take gas samples from greater depths, the sample tube used must be much more robust.

The levels of carbon monoxide, carbon dioxide and methane rose with increasing depth into the waste, while the oxygen level fell. The level of oxygen at a distance of 15m behind the face line was sufficient to sustain open fire. Extrapolating the fall in oxygen level with increasing distance, showed that the oxygen level did not fall below 17%, the level above which spontaneous combustion may be initiated (52), before a depth of 70m.

The benefit of monolithic packing over conventional packing methods in reducing the air quantity leaking through the waste was identified.

8.4 WASTE FLOW MODELLING

The pressure distribution in the waste produced by different waste permeabilities, boundary pressure conditions and nitrogen injection were examined.

When a constant permeability throughout the waste was used, a uniform pressure distribution was obtained, which gave airflow paths almost straight across the waste. This may happen in mines where the waste is not well consolidated.

The weak roof strata in most UK coal mines ensures that the waste is loaded, and consolidated by the strata pressure. This will produce a reduction in permeability from the perimeter of the waste towards the centre. Using this permeability arrangement, the pressure and velocity distributions indicated that little flow would occur straight across the waste. Most of the flow occurred at the perimeter of the waste. Air leaked through the pack into the unconsolidated corridor, where it flowed parallel to the airway. Towards the face end, the air moved diagonally across the face end towards the face. Airflow behind the face line was parallel to the face line. The flow pattern of the return side was a mirror image of that on the intake.

The pressure distribution produced by a fault running from the intake to the face differed little to that with no fault. The velocity distribution showed how air could flow along the fault to the face.

A point source of flow was used to simulate nitrogen injection. Injecting nitrogen into the permeable strata raised the pressure at the injection site, and resulted in a largely radial flow away from the source. Some nitrogen was forced back into the intake, a large fraction moved down the unconsolidated corridor to the face, while the balance moved through the waste to the face.

In the second set of exercises, a correlation was conducted between the quantity predicted to be leaking into the return, with the actual measured value. This resulted in the waste permeabilities being reduced to one third of the previous values. Subsequent exercises were conduced using this reduced permeability. Results from these exercises indicated flow patterns little different to those obtained earlier, albeit with lower velocities. The airflow in parts of the waste was identified as being in the non laminar flow regime.

The airflow pattern produced for a retreat face agreed with that of Grashchenkov (56).

A potential area at risk of spontaneous combustion in the waste of an advance face was produced.

8.5 FURTHER WORK

The one great assumption that has to be made when modelling waste flows is that of the permeability. In order for the waste flow velocity to be really accurately calculated, more precise values of permeability are needed. The permeabilities in the second

set of exercises were changed to correlate predicted leakage with that measured. Further work should be an extension of this, aimed towards getting actual permeability values for deep in the waste.

The key to this lies in developing a reliable method to obtain gas samples from deep in the waste. A more robust method of protecting the sample tube has to be used. Thick steel piping may provide an answer, however this would have to be extended every time the face advanced. Alternatively, a hole could be drilled to the required depth in the waste. Both these methods require greater financial and manpower resources than were available for this project.

Once it is possible to obtain gas samples from deep in the waste, a sampling program should be conducted. As well as gas samples, the pressure difference between the point in the waste and the roadway or face should be measured. Extensive airflow measurements would also need to be taken in the airways and on the face. In addition, the same measurements should be conducted on a face with nitrogen injection, as at Daw Mill in Chapter 5, with the injected nitrogen labelled using a tracer gas. This would identify the path of nitrogen from the injection site in the waste.

The information from such measurements may then be used to calculate the permeabilities in the waste, and correlate a finite element model with actual conditions.

Another method of finding the permeability in the waste might be a laboratory based experiment to investigate the permeability of broken coal and shale under increasing load. The author feels that it would be difficult to obtain realistic permeability values using this type of method.

A further consideration has to be the areas of non laminar flow. The method used to simulate this regime of flow needs to be considerably more sophisticated than the one that was used. Whether such sophistication is warranted is something that future workers will have to determine.

In addition to the work on spontaneous combustion, two additional areas that are affected by air movement through the waste are methane emission and the heat input to the ventilating air. At Harworth, the air leaking through the waste can contribute 40% of the total heat input to the air (60). Most of this occurs where air reenters the face at the return packside. The program used pressure differences as the potential causing flow. This is not strictly the case, it is the difference in total energy content that causes air to move. This may significantly alter the flow patterns in areas where the heat input is large. In gassy seams, a large leakage through the waste can result in methane concentrations in the return that exceed statutory levels, but little leakage can lead to an explosive atmosphere being formed near to the working areas.

A useful tool for the mining engineer would be the development of a computer method that is user friendly, which the program used here is not, to predict the leakage of air through the waste.

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REFERENCES

1. Plot R: Natural History of Staffordshire, 1686

2. Morris I H & Walker G 'Changes in the Approach to Ventilation in Recent Years'
The Mining Engineer, January 1982, V141, p401

3. Richters E: 'A Peculiar Change that Coal Undergoes on Heating in Air'
Dingler's Polytechnic Journal, vol190, P398

4. Winmill T F: 'Absorbtion of Oxygen by Coal'
The Mining Engineer, vol 48, P508

5. Guney M: 'Oxidation and Spontaneous Combustion of Coal, A Review of Individual Factors'
Colliery Guardian, V216, p105

6. Hodges D J & Acherjee B: 'A Microcalorimetric Study of the Influence of Moisture on the Spontaneous Heating of Coal'
The Mining Engineer, V126, p121

7. Banerjee S C: 'Spontaneous Combustion of Coal and Mine Fires'
pub A A Balkema, Rotterdam 1985

8. Chamberlain E A: 'The Ambient Temperature Oxidation of Coal in Relation to the Early Detection of Spontaneous Combustion '
The Mining Engineer, V132, p387

9. Schmit J L et al: 'Relative Spontaneous Heating Tendency of Coals'
U.S.B.M., 1945, paper 681

10. Bouwman R & W Freiks J: 'Low Temperature Oxidation of a Bituminous Coal'
Fuel, V59, p315

11. Kam A, Hixon A & Perlmutter D: 'The Oxidation of Bituminous Coal'
Chemical Engineering Science, 1976, V31, p815

12. Gan S et al: 'Nature of the Porosity in American Coals'
Fuel, Oct 1972, V51, p272

13. Smith S E et al: 'DTGA Combustion of Coal in the Exxon Coal Laboratory'
Fuel, V60, p458

14. Acherjee B A: 'Spontaneous Heatings'
Colliery Guardian, 1971, V219, p49

15. Bannerjee S C et al: 'Rate Studies of Aerial Oxidation of Coal at Low Temperatures'
Fuel, V49, p324

16. Demirbilek S: 'The Development of a Spontaneous Combustion Risk Classification System for Coal Seams'
University of Nottingham, PhD thesis, 1986

17. Hodges D J & Guney M: 'Adiabatic Studies of the Spontaneous Heating of Coal'
Colliery Guardian, V217, p173

18. Verma Y K: 'Underground Virgin Strata Temperature Measurements in the Penine Coalfields'
The Mining Engineer, August 1979, V139, p169

19. Collier L: Presidential Address to the Institution of Mining Engineers South Staffs. & S. Midlands
The Mining Engineer, January 1985, V144

20. Carter W H & Durst C S: 'The Influence of Barometric Changes on the Emission of Firedamp'
The Mining Engineer, V115, p3

21. Jolliffe J V & Raybould W E: 'The Application of Pressure Balancing Chambers to Control Movement in Sealed Areas'
The Mining Engineer, August 1961, V120, p861

22. Faucconnier C J & Buekes J D:'Some Interesting Observations About Colliery Fires and the Weather'
Journal Mine Ventilation Society of South Africa, April 1978, p57

23. Faucconnier C J & Buekes J D: 'The Determination of Optimal Gas Sampling Times'
Journal Mine Ventilation Society of South Africa, V33, P169

24. Guney M: 'Equilibrium Humidities and Water Isotherms in Relation to Spontaneous Heating of Coals'
2nd International Mine Ventilation Congress, Johannesburg 1975

25. Hodges D J, Hinsley F & Guney M: 'An Investigation of the Spontaneous Heating of Coal and Gaseous Products'
The Mining Engineer, 1969, V129, p67

26. Hodges D J, Hinsley F & Bhattacharyya B: 'The Influence of Humidity on the Initial Stages of the Spontaneous Heating of Coal'
The Mining Engineer, 1969, V128, p274

27. Hill D J: 'A Spontaneous Heating Assessment of Longwall Mining Operations'
BSc Thesis, University of Nottingham, 1984

28. Ventilation in Coal Mines
pub NCB Mining Dept, 1979

29. Wastell E R & Walker G: 'The Use of Nitrogen at Fryston Colliery'
The Mining Engineer, V143, No262, p27

30. Lord S B: 'Some Aspects of Spontaneous Combustion Control'
The Mining Engineer, May 1986, V145

31. NCB Western Area, Mining Department Instruction 19, Spontaneous Combustion, Revised 1982

32. Highton W: 'Spontaneous Combustion, Its Early Detection and Latest Means of Combat in the Western Area' The Mining Engineer, July 1982, V142, p43

33. Lewis S & Stace R: 'A Rational of Roadside Packing' The Mining Engineer, V140, p717

34. Bexon R: 'Monolithic Packing' The Mining Engineer, May 1986, V145

35. 'Evaluation of Thermal Imagers for the Detection of Fires and Heatings Underground' NCB South Midlands Area/ECSC Research Project No7255 - 10/035/08

36. Humphreys D J: 'Evaluation of a Tube Bundle System for Goaf Atmosphere Monitoring and Detection of Spontaneous Combustion' Australian Coal Research Laboratories, Report 82-10

37. Criddle S J: 'Developments in Mining Science, With Reference to the Detection of Spontaneous Combustion' The Mining Engineer, V141, No238, p47

38. Eicker H: 'Investigation of Methods for the Early Detection of Fires Due to Spontaneous Combustion' 3rd Mining Ventilation Congress, Harrogate 1984

39. Childs D T: 'Spontaneous Combustion at Daw Mill'
Paper presented to the S.Staffs and Midlands Branch Institution of Mining Engineers, 17 December 1984

40. Brandle I E: 'Successful Counter Action Against a Build up of CO by Means of Pressure Balancing'
Gluckauf, V116, No2, p25

41. Harris L: 'The Use of Nitrogen to Control Spontaneous Combustion Heatings'
The Mining Engineer, June 1981, V140, p883

42. Frycz F: 'Firefighting in High Methane Seams in Polish Coal Mines'
3rd International Mine Ventilation Congress, Harrogate 1984

43. Drake D & Highton W: 'Financial Aspects of Spontaneous Combustion'
The Mining Engineer, July 1986

44. Zillessen A: 'A New Air Separation Unit for the Production of Inert Gas'
Gluckauf, V119, No16, p305

45. Vaughan-Thomas T: 'The Use of Nitrogen in Controlling an Underground Fire at Fernhill Colliery'
The Mining Engineer, March 1964, V123, p311

46. Both W: 'Fighting Mine Fires with Nitrogen in the German Coal Industry'
Gluckauf, V115, No19, p41

47. Skelding M E: 'Heavy Duty Chock Salvage With and Without Nitrogen'
The Mining Engineer, August 1983, V143, p71

48. Pickering A: 'A Study of Air Flows and Methane Emissions on Fully Caved Mechanised Wastes in the South Nottinghamshire Area'
University of Nottingham, MSc Thesis, 1965

49. Clough R & Carver J: 'A Study of the Methane/Air Mixtures In The Wastes of Longwall Workings'
Transactions of the Institute of Mining Engineers, 1956, V116, p131

50. Stokes A: 'Air Leakage Through Longwall Wastes in the Sydney Coalfield'
2nd US Mine Ventilation Symposium, Reno, Sept 1985

51. Freidman S: 'Methane Concentration in the Worked Out Space'
Soviet Mining Science, May - June 1984, V20, No 3

52. Evseev V: 'Prevention of Spontaneous Heating During Extraction of Thick Seam in Kuzbass'
20th International Conference of Safety in Mines Research Institutes, Sheffield, October 1983

53. Barends R & Hannoura F: 'Non Darcy Flow. A State of The Art'
proc. Euromech 143, Delft, Sept 1981, pub A A Balkema, Rotterdam, ISBN 90 6191 216 4

54. 'Flow Through Porous Media'
ed R J de Wiest, pub Academic Press, New York and London, 1969

55. Brunner D J: 'Ventilation Models for Longwall Gob Leakage Simulation'
2nd US Mine Ventilation Symposium, Reno, September 1985

56. Grashchenkov N F et al: 'Air Loss Through the Worked Out Space'
Soviet Mining Science, Vol 20, No 3, May-June 1984

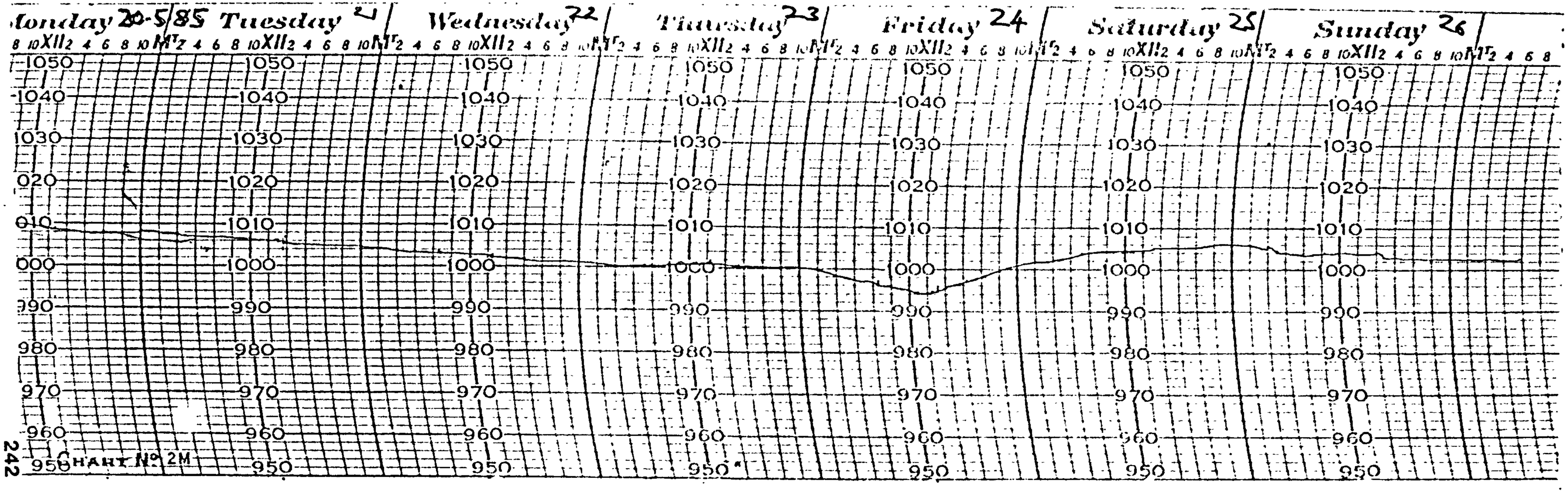
57. Segerlind L J: 'Applied Finite Element Analysis'
John Wiley and Sons, ISBN 0 471 80662 5

58. Whittaker B N: 'An Appraisal of Strata Control Practice'
The Mining Engineer, Vol 134, p9

59. Richards M J, Nottingham University, Private Communication

60. Tuck M A: 'Computer Simulation of Climate on a Longwall Face'
University of Nottingham, PhD. Thesis, 1986

APPENDIX 1
BAROMETRIC TRACE AT HUCKNALL COLLIERY, AND ALL ANALYSES OF GAS SAMPLES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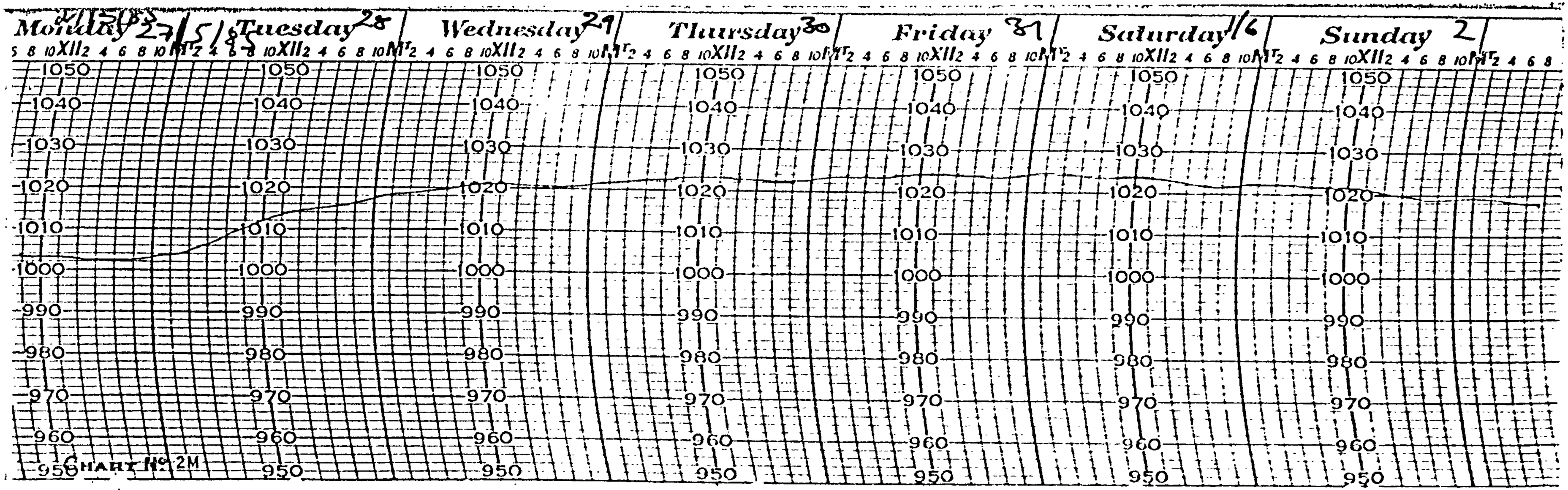
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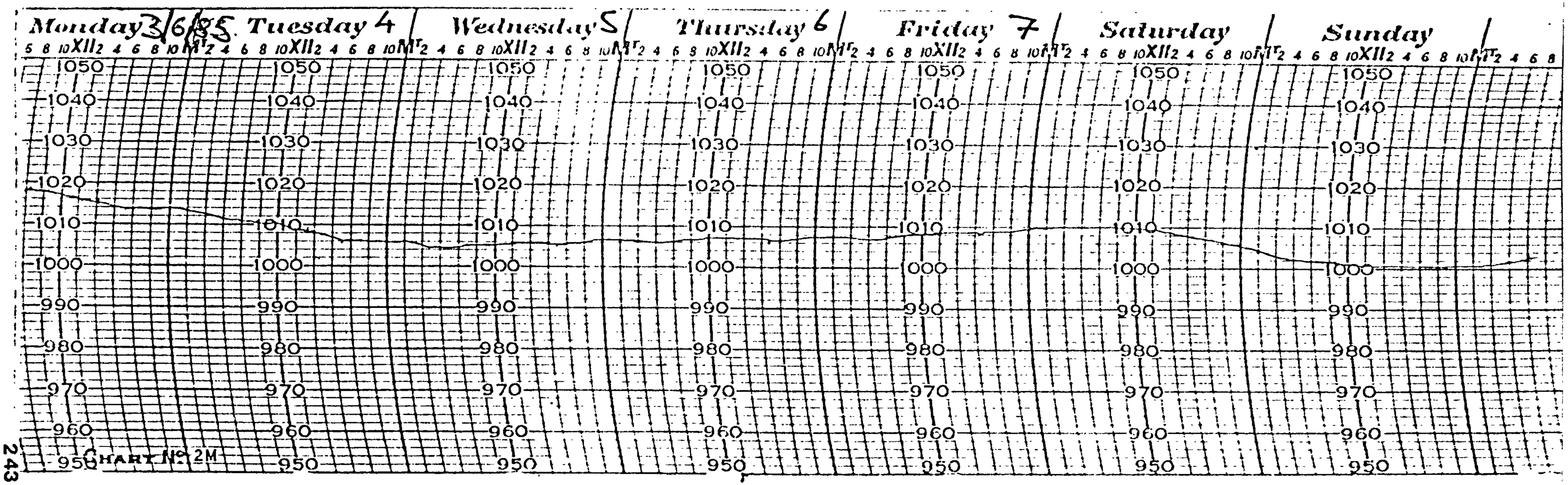
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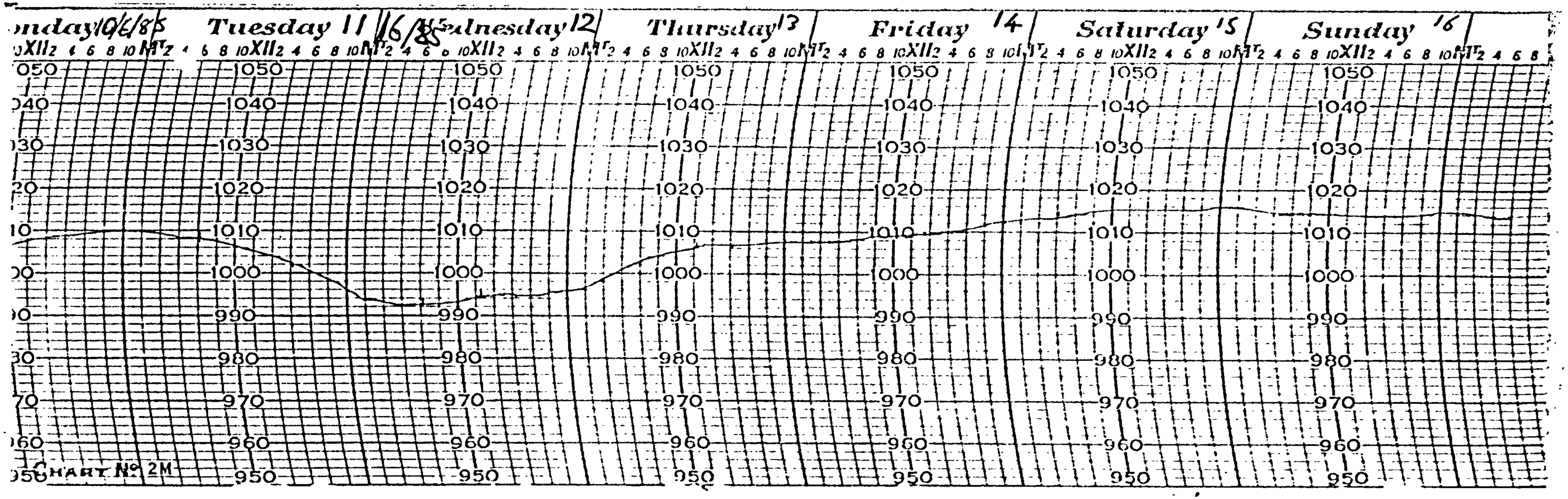
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CHART No. 2M

19/8/85

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CHART No 2M

244

26/8/85

Monday 26 | Tuesday 27 | Wednesday 28 | Thursday 29 | Friday 30 | Saturday 31 | Sunday 1

8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8

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CHART No. 2M

245

Monday 9 2/25 Tuesday 10 | Wednesday 11 | Thursday 12 | Friday 13 | Saturday 14 | Sunday 15

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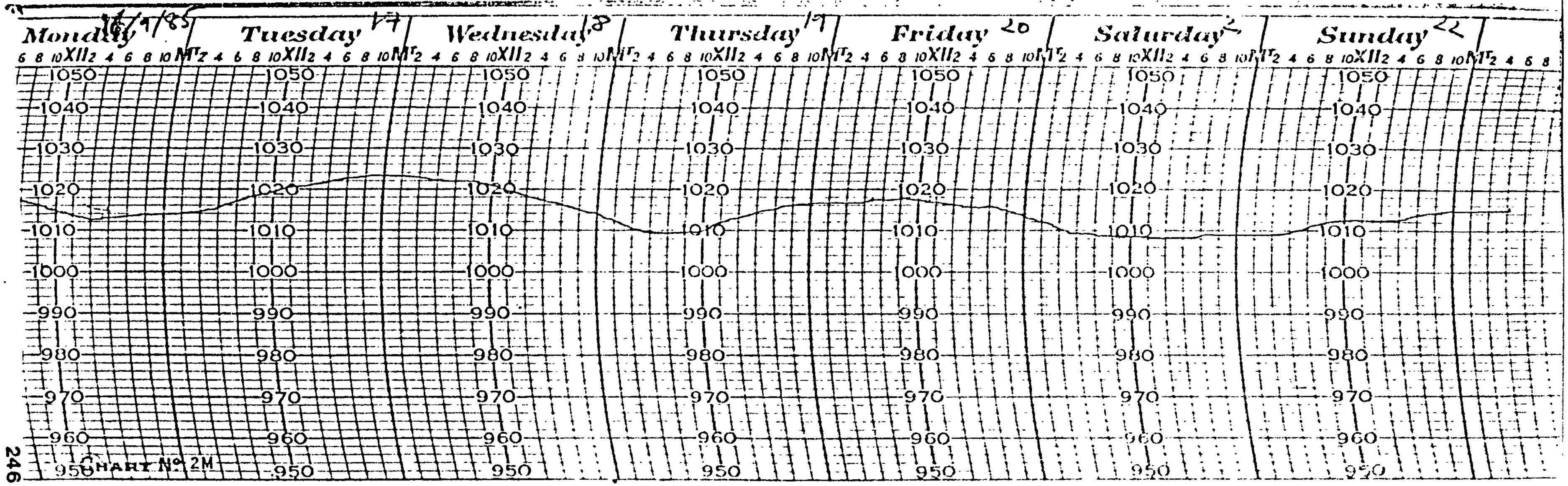
CHART No. 2M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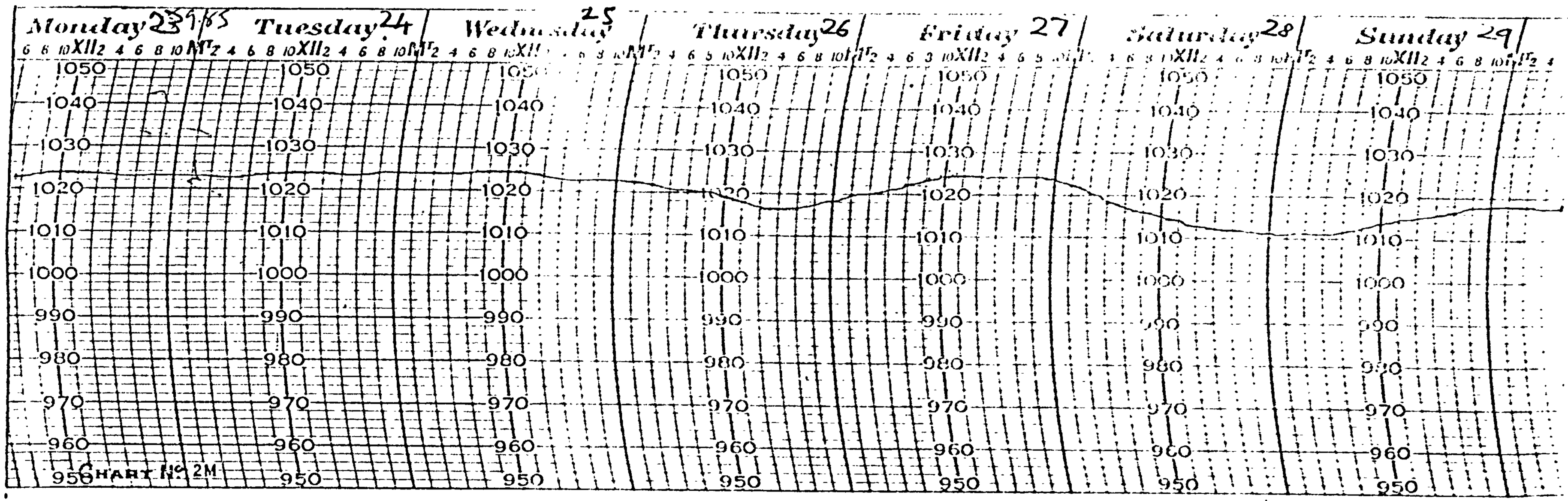
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Saturday 28
Sunday 29
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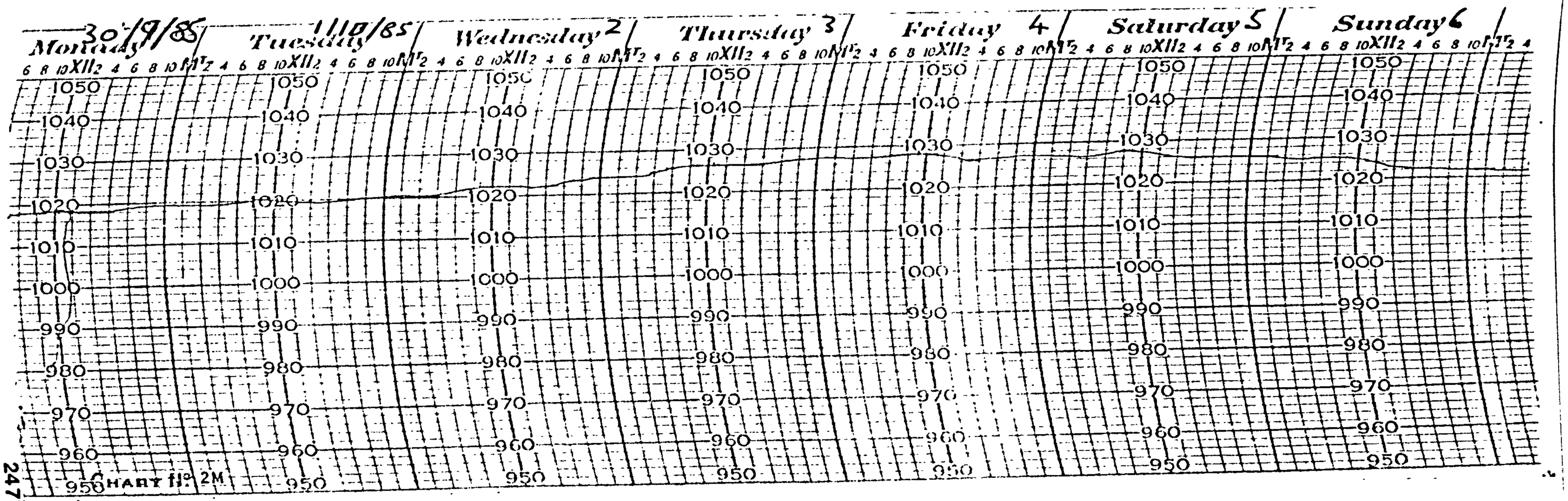
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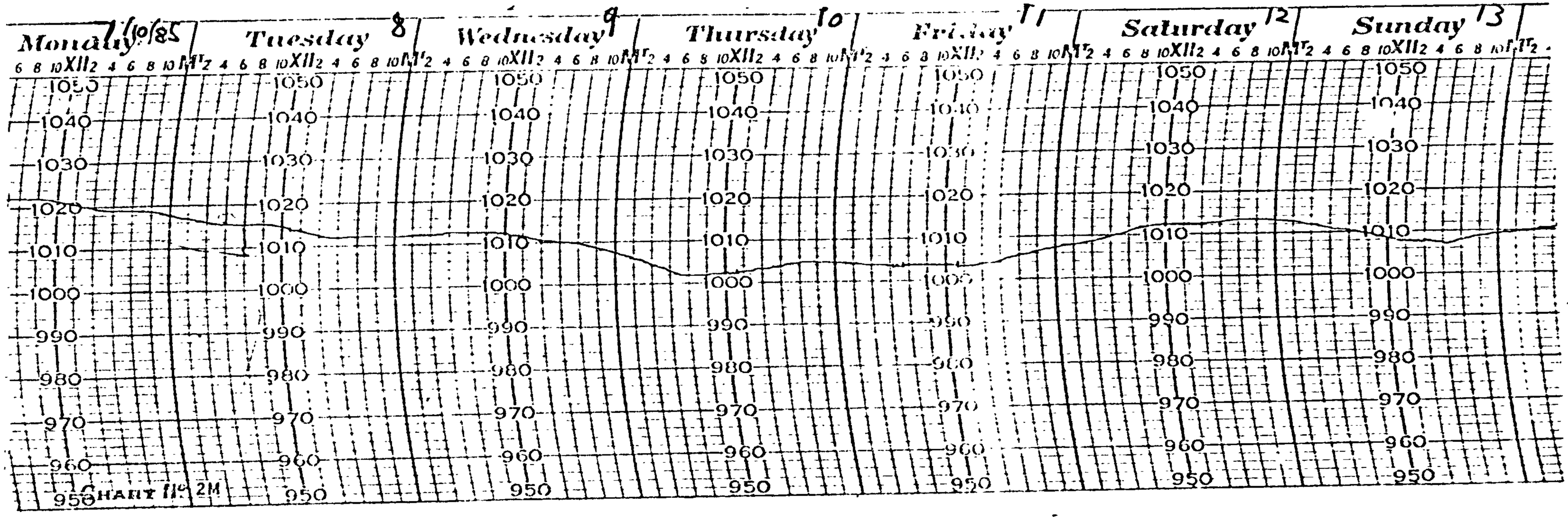
7/10/85
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CHART No. 2M

Monday 14/10/85 | Tuesday 15 | Wednesday 16 | Thursday 17 | Friday 18 | Saturday 19 | Sunday 20

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CHART No 2M

Monday 21/10/85 | Tuesday 22 | Wednesday 23 | Thursday 24 | Friday 25 | Saturday 26 | Sunday 27

6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8 10 XII 2 4 6 8 10 MT 2 4 6 8

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CHART No 2M

Monday 29/10/85 Tuesday 29 Wednesday 30 Thursday 31 Friday 1/11/85 Saturday 2 Sunday 3

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CHART No. 2M

Monday 4/11/85 Tuesday 5 Wednesday 6 Thursday 7 Friday 8 Saturday 9 Sunday 10

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CHART No. 2M

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
16/8/85	OBE RETURN	0.13	0.26	20.60	79.01	0.0008	0.32	0.25
	FACE GB, 40 CHOCK	0.23	0.53	19.95	79.71	0.0039	1.57	0.25
	GB INTAKE	0.07	0.05	20.80	79.08	0.0003	0.14	0.21
19/8	OBE RETURN			NO SAMPLE				
	10M RETURN RIP	0.15	0.34	20.57	78.94	0.0007	0.33	0.21
	FACE GB, 40 CHOCK	0.14	0.27	20.62	78.97	0.0004	0.29	0.14
	INTAKE	0.11	0.20	20.69	79.00	0.0004	0.23	0.17
20/8	OBE RETURN	0.13	0.25	20.59	79.03	0.0007	0.34	0.21
	10M RETURN RIP	0.11	0.22	20.64	79.03	0.0006	0.29	0.21
	FACE GB, 40 CHOCK	0.09	0.15	20.70	79.06	0.0004	0.23	0.17
	INTAKE	0.09	0.15	20.72	79.04	0.0003	0.21	0.14

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
21/8	OBE RETURN	0.13	0.28	20.62	78.97	0.0008	0.29	0.28
	10M RETURN RIP	0.09	0.18	20.74	78.99	0.0006	0.18	0.33
	FACE GB, 40 CHOCK	0.07	0.07	20.78	79.08	0.0004	0.16	0.25
	WASTE SAMPLES							
16/8	FACE ADVANCE 0.5M	0.08	0.10	20.76	79.06	0.0004	0.17	0.27
19/8	FACE ADVANCE 1.7M	0.16	0.47	20.37	79.00	0.0016	0.55	0.29
20/8	FACE ADVANCE 3.0M	0.14	0.40	20.27	79.19	0.0024	0.70	0.34
21/8	FACE ADVANCE 4.0M	0.11	0.29	20.52	79.08	0.0014	0.42	0.33

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
23/8	OBE RETURN	0.11	0.41	20.49	78.99	0.0011	0.43	0.26
	10M RETURN RIP	0.08	0.35	20.60	78.97	0.0010	0.31	0.32
	FACE GB	0.08	0.21	20.67	79.04	0.0007	0.26	0.27
	INTAKE	0.08	0.16	20.72	79.04	0.0007	0.21	0.33
29/8	OBE RETURN	0.08	0.15	20.64	79.13	0.0008	0.31	0.26
	10M RETURN RIP	0.06	0.11	20.73	79.10	0.0006	0.21	0.29
	FACE GB	0.06	0.10	20.74	79.10	0.0006	0.20	0.30
	INTAKE	0.05	0.04	20.81	79.10	0.0005	0.13	0.38
30/8	OBE RETURN	0.10	0.25	20.57	79.05	0.0011	0.37	0.30
	10M RETURN RIP	0.09	0.24	20.64	79.03	0.0008	0.29	0.28
	FACE GB	0.08	0.20	20.66	79.06	0.0008	0.27	0.30
	INTAKE	0.07	0.11	20.72	79.10	0.0005	0.22	0.23

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
02/10	OBE RETURN	0.12	0.25	20.58	79.05	0.0009	0.35	0.26
	10M RETURN RIP	0.10	0.22	20.66	79.02	0.0007	0.26	0.27
	FACE GB	0.08	0.19	20.71	79.02	0.0004	0.21	0.19
	INTAKE	0.06	0.11	20.77	79.06	0.0003	0.16	0.19
03/10	OBE RETURN	0.14	0.29	20.57	79.00	0.0010	0.35	0.29
	10M RETURN RIP	0.12	0.26	20.66	78.96	0.0009	0.25	0.30
	FACE GB	0.10	0.24	20.67	78.99	0.0006	0.25	0.24
	INTAKE	0.08	0.14	20.73	79.05	0.0004	0.20	0.20
	WASTE SAMPLES							
23/8	FACE ADVANCE 6.7M	0.09	0.30	20.30	79.31	0.0024	0.80	0.30
29/8	FACE ADVANCE 0.5M	0.06	0.21	20.63	79.00	0.0008	0.29	0.28
30/8	FACE ADVANCE 1.7M	0.11	0.38	20.29	79.22	0.0020	0.68	0.29
02/9	FACE ADVANCE 3.5M	0.09	0.26	20.45	79.20	0.0013	0.52	0.25
03/9	FACE ADVANCE 4.2M	0.17	0.54	20.12	78.17	0.0024	0.55	0.41

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
04/9	OBE RETURN	0.11	0.17	20.66	79.01	0.0007	0.27	0.26
	10M RETURN RIP	0.08	0.11	20.75	79.06	0.0005	0.18	0.28
	FACE GB	INSUFFICIENT SAMPLE						
	INTAKE	0.05	0.03	20.83	79.09	0.0003	0.11	0.27
05/9	OBE RETURN	0.13	0.31	20.55	79.01	0.0008	0.37	0.22
	10M RETURN RIP	0.09	0.19	20.68	79.04	0.0005	0.25	0.20
	FACE GB	0.09	0.13	20.73	79.05	0.0004	0.20	0.20
	INTAKE	0.07	0.05	20.81	79.07	0.0003	0.12	0.25
	WASTE SAMPLES							
04/9	FACE ADVANCE 5.1M	0.13	0.31	20.37	79.19	0.0020	0.60	0.33
04/9	FACE ADVANCE 5.1M	0.13	0.31	20.35	79.21	0.0021	0.62	0.34
05/9	FACE ADVANCE 6.2M	0.15	0.41	20.27	79.17	0.0019	0.69	0.28

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
06/9	OBE RETURN	0.07	0.22	20.63	79.05	0.0008	0.31	0.26
	10M RETURN RIP	0.04	0.11	20.74	79.11	0.0006	0.20	0.30
	FACE GB	0.04	0.08	20.75	79.13	0.0005	0.20	0.25
	INTAKE	0.03	0.02	20.84	79.11	0.0003	0.10	0.30
09/9	OBE RETURN	0.08	0.23	20.63	79.06	0.0007	0.30	0.23
	10M RETURN RIP	0.05	0.11	20.76	79.08	0.0004	0.18	0.22
	FACE GB	0.04	0.08	20.77	79.11	0.0003	0.17	0.18
	INTAKE	0.04	0.04	20.83	79.09	0.0003	0.11	0.27
10/9	OBE RETURN	0.08	0.21	20.64	79.07	0.0007	0.29	0.24
	10M RETURN RIP	0.05	0.13	20.73	79.09	0.0004	0.21	0.19
	FACE GB	0.05	0.10	20.75	79.10	0.0004	0.19	0.27
	INTAKE	0.04	0.04	20.83	79.09	0.0003	0.11	0.27

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
06/9	FACE ADVANCE 7M	0.10	0.35	20.24	79.28	0.0023	0.75	0.31
09/9	FACE ADVANCE 8.2M	0.09	0.29	20.36	79.26	0.0019	0.63	0.30
10/9	FACE ADVANCE 9.0M	0.14	0.42	20.09	79.35	0.0031	0.92	0.34

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
11/9	OBE RETURN	0.12	0.18	20.64	79.06	0.0008	0.29	0.28
	10M RETURN RIP	0.07	0.11	20.76	79.06	0.0004	0.17	0.24
	FACE GB	0.07	0.09	20.75	79.05	0.0005	0.19	0.26
	INTAKE	0.06	0.03	20.79	79.12	0.0004	0.15	0.27
12/9	OBE RETURN	0.16	0.32	20.51	79.01	0.0010	0.41	0.24
	10M RETURN RIP	0.11	0.24	20.60	79.05	0.0007	0.33	0.21
	FACE GB	0.11	0.21	20.62	79.06	0.0007	0.31	0.23
	INTAKE	0.11	0.15	20.74	79.00	0.0006	0.18	N/A
13/9	OBE RETURN	0.13	0.27	20.61	78.99	0.0007	0.31	0.23
	10M RETURN RIP	0.08	0.17	20.74	79.01	0.0005	0.18	0.28
	FACE GB	0.07	0.10	20.81	79.02	0.0004	0.11	N/A

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
11/9	FACE ADVANCE 0.0M	0.08	0.10	20.72	79.10	0.0006	0.22	0.27
12/9	FACE ADVANCE 0.5M	0.12	0.22	20.62	79.04	0.0008	0.31	0.26
13/9	FACE ADVANCE 1.5M	0.09	0.15	20.66	79.10	0.0007	0.28	0.25

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
16/9	OBE RETURN	0.12	0.24	20.60	79.04	0.0008	0.33	0.24
	10M RETURN RIP	0.14	0.18	20.70	78.98	0.0005	0.22	0.23
	FACE GB	0.08	0.15	20.72	79.05	0.0004	0.21	0.19
	INTAKE	0.12	0.12	20.75	79.01	0.0003	0.17	0.18
17/9	OBE RETURN	0.12	0.19	20.63	79.06	0.0008	0.30	0.26
	10M RETURN RIP	0.08	0.17	20.73	79.02	0.0004	0.19	0.21
	FACE GB	0.08	0.14	20.74	79.04	0.0004	0.19	0.21
	INTAKE	0.06	0.05	20.81	79.08	0.0003	0.13	0.23
18/9	OBE RETURN	0.12	0.24	20.60	79.04	0.0008	0.33	0.24
	10M RETURN RIP	0.10	0.20	20.72	78.98	0.0006	0.20	0.30
	FACE GB	0.12	0.14	20.74	79.00	0.0004	0.18	0.22
	INTAKE	0.10	0.08	20.78	79.04	0.0004	0.15	0.27

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
19/9	OBE RETURN	0.18	0.38	20.50	78.94	0.0008	0.40	0.20
	10M RETURN RIP	0.12	0.28	20.65	78.95	0.0004	0.26	0.15
	FACE GB	0.16	0.32	20.58	78.94	0.0007	0.32	0.22
	INTAKE	0.14	0.15	20.73	78.98	0.0004	0.19	0.21
	WASTE SAMPLES							
16/9	FACE ADVANCE 2.5M	0.12	0.23	20.63	79.02	0.0007	0.29	0.24
17/9	FACE ADVANCE 3.5M	0.16	0.38	20.52	78.94	0.0012	0.38	0.32
18/9	FACE ADVANCE 4.5M	0.12	0.26	20.53	79.09	0.0012	0.41	0.30
19/9	FACE ADVANCE 0.5M	0.16	0.27	20.65	78.92	0.0005	0.25	0.20

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
20/9	OBE RETURN	0.10	0.23	20.57	79.10	0.0009	0.37	0.24
	10M RETURN RIP	0.08	0.18	20.64	79.10	0.0008	0.30	0.27
	FACE GB	0.07	0.16	20.72	79.05	0.0005	0.21	0.24
23/9	OBE RETURN	0.12	0.19	20.62	79.07	0.0008	0.31	0.26
	10M RETURN RIP	0.06	0.12	20.75	79.07	0.0005	0.18	0.28
	FACE GB	0.06	0.11	20.74	79.09	0.0005	0.20	0.25
	INTAKE	0.06	0.05	20.79	79.10	0.0004	0.15	0.27
24/9	OBE RETURN	0.10	0.20	20.62	79.08	0.0008	0.32	0.25
	10M RETURN RIP	0.06	0.12	20.72	79.10	0.0005	0.22	0.23
	FACE GB	0.06	0.10	20.72	79.12	0.0005	0.23	0.22
	INTAKE	0.03	0.03	20.81	79.13	0.0004	0.14	0.29

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
20/9	FACE ADVANCE 1.0M	0.08	0.16	20.68	79.08	0.0007	0.26	0.27
23/9	FACE ADVANCE 2.0M	0.07	0.11	20.73	79.09	0.0006	0.21	0.29
24/9	FACE ADVANCE 2.5M	0.20	0.13	19.56	79.11	0.0025	1.38	0.18

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
25/9	OBE RETURN	0.10	0.16	20.64	79.10	0.0007	0.30	0.23
	10M RETURN RIP	0.08	0.14	20.69	79.09	0.0006	0.25	0.24
	FACE GB	0.06	0.10	20.74	79.10	0.0005	0.20	0.25
	INTAKE	0.05	0.04	20.80	79.11	0.0004	0.14	0.29
	WASTE SAMPLES							
25/9	FACE ADVANCE 3.0M	0.10	0.20	20.52	79.18	0.0014	0.45	0.31

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
26/9	OBE RETURN	0.13	0.15	20.60	79.12	0.0009	0.35	0.26
	10M RETURN RIP	0.08	0.12	20.71	79.09	0.0006	0.23	0.26
	FACE GB	0.10	0.16	20.71	79.03	0.0006	0.22	0.27
	INTAKE	INSUFFICIENT SAMPLE						
27/9	OBE RETURN	0.16	0.24	20.56	79.04	0.0009	0.37	0.24
	10M RETURN RIP	0.09	0.14	20.68	79.09	0.0006	0.26	0.23
	FACE GB	0.10	0.16	20.71	79.03	0.0006	0.22	0.27
	INTAKE	0.08	0.03	20.77	79.12	0.0006	0.18	N/A
30/9	OBE RETURN	0.17	0.25	20.58	79.00	0.0007	0.34	0.21
	10M RETURN RIP	0.12	0.18	20.68	79.02	0.0005	0.24	0.21
	FACE GB	0.12	0.14	20.71	79.03	0.0004	0.22	0.18
	INTAKE	INSUFFICIENT SAMPLE						

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
26/9	FACE ADVANCE 3.7M	0.12	0.20	20.48	79.20	0.0016	0.49	0.33
29/9	FACE ADVANCE 4.75M	0.16	0.21	20.41	79.22	0.0018	0.56	0.32
30/9	FACE ADVANCE 6.0M	0.12	0.16	20.65	79.07	0.0006	0.28	0.21

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
01/10	OBE RETURN	0.17	0.29	20.54	79.00	0.0008	0.38	0.21
	10M RETURN RIP	0.14	0.24	20.65	78.97	0.0006	0.26	0.23
	FACE GB	0.12	0.22	20.68	78.98	0.0005	0.24	0.21
02/10	OBE RETURN	0.16	0.26	20.56	79.02	0.0008	0.36	0.22
	FACE GB	0.11	0.20	20.71	78.98	0.0005	0.21	0.24
	INTAKE	0.10	0.10	20.77	78.98	0.0004	0.16	0.25
03/10	OBE RETURN	0.25	0.54	20.38	78.83	0.0008	0.50	0.16
	10M RETURN RIP	0.20	0.42	20.50	78.88	0.0005	0.39	0.13
	FACE GB	0.18	0.38	20.54	78.90	0.0005	0.35	0.14
	INTAKE	0.16	0.28	20.64	78.94	0.0004	0.28	0.14

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
01/10	FACE ADVANCE = 2.0	0.20	0.42	20.27	79.11	0.0017	0.67	0.25
02/10	FACE ADVANCE = 3.5	0.14	0.34	20.46	79.06	0.0014	0.47	0.30
03/10	FACE ADVANCE = 5.0	0.22	0.56	20.30	78.92	0.0012	0.60	0.20

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
04/10	OBE RETURN	0.15	0.31	20.54	79.00	0.0007	0.38	0.18
	10M RETURN RIP	0.11	0.27	20.65	78.97	0.0005	0.26	0.19
	FACE GB	0.09	0.23	20.66	79.02	0.0004	0.27	0.15
	INTAKE	0.09	0.17	20.74	79.00	0.0003	0.18	0.17
07/10	OBE RETURN	0.11	0.21	20.63	79.05	0.0007	0.30	0.23
	10M RETURN RIP	0.09	0.19	20.66	79.06	0.0006	0.27	0.22
	FACE GB	0.07	0.13	20.75	79.05	0.0004	0.18	0.22
	INTAKE	0.05	0.05	20.82	79.08	0.0003	0.11	0.27
08/10	OBE RETURN	0.11	0.18	20.63	79.08	0.0007	0.30	0.23
	10M RETURN RIP	0.08	0.17	20.71	79.04	0.0005	0.22	0.23
	FACE GB	0.07	0.13	20.76	79.04	0.0003	0.17	0.18

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
09/10	OBE RETURN	0.13	0.23	20.60	79.04	0.0007	0.33	0.21
	10M RETURN RIP	0.11	0.21	20.66	79.02	0.0006	0.27	0.22
	FACE GB	0.09	0.15	20.74	79.02	0.0003	0.19	0.16
	WASTE SAMPLES							
04/10	FACE ADVANCE = 5.75	0.14	0.40	20.40	79.06	0.0013	0.53	0.25
07/10	FACE ADVANCE = 6.5	0.11	0.31	20.44	79.14	0.0014	0.51	0.27
08/10	FACE ADVANCE = 0.0	0.07	0.15	20.68	79.10	0.0006	0.26	0.23
09/10	FACE ADVANCE = 1.25	0.04	0.21	20.37	79.38	0.0019	0.64	0.30

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
10/10	OBE RETURN	0.10	0.16	20.60	79.14	0.0007	0.35	0.20
	10M RETURN RIP	0.06	0.16	20.69	79.11	0.0005	0.29	0.19
	FACE GB	0.06	0.12	20.72	79.10	0.0005	0.22	0.23
	INTAKE	0.05	0.04	20.76	79.15	0.0003	0.20	0.15
11/10	OBE RETURN	0.10	0.15	20.62	79.12	0.0009	0.33	0.21
	10M RETURN RIP	0.09	0.18	20.90	79.05	0.0004	0.23	0.19
	FACE GB	0.06	0.15	20.74	79.05	0.0003	0.19	0.16
14/10	OBE RETURN	0.12	0.16	20.60	79.12	0.0021	0.35	0.60
	10M RETURN RIP	0.08	0.12	20.67	79.13	0.0010	0.28	0.36
	FACE GB	0.03	0.09	20.44	79.44	0.0016	0.59	0.27

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
15/10	OBE RETURN	0.12	0.24	20.64	79.10	0.0010	0.40	0.25
	10M RETURN RIP	0.08	0.20	20.65	79.07	0.0008	0.28	0.29
	FACE GB	0.07	0.10	20.72	79.11	0.0006	0.22	0.29
	WASTE SAMPLES							
10/10	FACE ADVANCE = 1.75	0.07	0.16	20.66	79.11	0.0006	0.28	0.21
11/10	FACE ADVANCE = 2.6	0.07	0.20	20.63	79.10	0.0006	0.31	0.19
	FACE ADVANCE = 2.6	0.06	0.18	20.64	79.12	0.0006	0.31	0.19
14/10	FACE ADVANCE = 3.5	0.10	0.22	20.52	79.16	0.0015	0.44	0.34
	FACE ADVANCE = 3.5	0.08	0.22	20.51	79.19	0.0015	0.46	0.33
15/10	FACE ADVANCE = 4.5	0.11	0.50	20.35	79.04	0.0017	0.58	0.29
	FACE ADVANCE = 4.5	0.10	0.32	20.40	79.18	0.0018	0.57	0.39

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
16/10	OBE RETURN	0.14	0.18	20.62	79.06	0.0008	0.29	0.28
	10M RETURN RIP	0.08	0.14	20.90	79.08	0.0005	0.24	0.21
	FACE GB	0.08	0.13	20.72	79.07	0.0004	0.21	0.19
	WASTE SAMPLES							
16/10	FACE ADVANCE = 5.0	0.14	0.59	20.30	78.97	0.0018	0.61	0.30
	FACE ADVANCE = 5.0	0.14	0.52	20.33	79.01	0.0018	0.59	0.31

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
17/10	OBE RETURN	0.13	0.21	20.60	79.06	0.0008	0.33	0.24
	10M RETURN RIP	0.10	0.18	20.70	79.02	0.0006	0.22	0.27
	FACE GB	0.08	0.15	20.72	79.05	0.0005	0.21	0.24
18/10	OBE RETURN	0.14	0.28	20.57	79.01	0.0008	0.35	0.23
	10M RETURN RIP	0.10	2.26	20.66	78.98	0.0006	0.26	0.23
	FACE GB	0.08	0.18	20.71	79.03	0.0005	0.22	0.23
21/10	OBE RETURN	0.13	0.16	20.62	79.09	0.0008	0.32	0.25
	10M RETURN RIP	0.08	0.12	20.73	79.07	0.0005	0.20	0.25
	FACE GB	0.08	0.10	20.76	79.06	0.0004	0.17	0.24

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
17/10	FACE ADVANCE = 5.75	0.14	0.55	20.27	79.04	0.0020	0.66	0.30
	FACE ADVANCE = 5.75	0.10	0.56	20.29	79.04	0.0020	0.66	0.30
18/10	FACE ADVANCE = 6.5	0.15	0.43	20.23	79.19	0.0023	0.74	0.31
	FACE ADVANCE = 6.5	0.15	0.44	20.23	79.18	0.0024	0.74	0.32
21/10	FACE ADVANCE = 7.6	0.13	0.32	20.31	79.24	0.0022	0.67	0.33
	FACE ADVANCE = 7.6	0.06	0.32	20.17	79.24	0.0024	0.87	0.28

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
22/10	OBE RETURN	0.13	0.13	20.67	79.07	0.0007	0.26	0.27
	10M RETURN RIP	0.09	0.10	20.73	79.08	0.0005	0.21	0.24
	FACE GB	0.09	0.07	20.78	79.06	0.0003	0.15	0.20
23/10	OBE RETURN	0.12	0.14	20.63	79.11	0.0006	0.31	0.19
	10M RETURN RIP	0.09	0.13	20.74	79.04	0.0004	0.19	0.21
	FACE GB	0.09	0.09	20.76	79.06	0.0004	0.17	0.24
28/10	OBE RETURN	0.15	0.17	20.61	79.07	0.0009	0.32	0.28
	10M RETURN RIP	0.12	0.15	20.69	79.04	0.0006	0.24	0.25
	FACE GB	0.10	0.14	20.59	79.03	0.0006	0.21	0.29
29/10	OBE RETURN	0.15	0.23	20.61	79.03	0.0007	0.34	0.21
	10M RETURN RIP	0.06	0.14	20.74	79.06	0.0007	0.19	0.37
	FACE GB	0.08	0.11	20.75	79.06	0.0003	0.18	0.17

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
22/10	FACE ADVANCE = 8.1	0.13	0.27	20.43	79.17	0.0020	0.53	0.38
	FACE ADVANCE = 8.1	0.13	0.27	20.44	79.16	0.0019	0.52	0.37
23/10	FACE ADVANCE = 9.5	0.15	0.38	20.32	79.15	0.0023	0.64	0.36
	FACE ADVANCE = 9.5	0.15	0.38	20.32	79.15	0.0023	0.64	0.36
28/10	FACE ADVANCE = 9.7	0.19	0.42	20.16	79.23	0.0033	0.82	0.40
	FACE ADVANCE = 9.7	0.19	0.42	20.18	79.21	0.0032	0.79	0.41
29/10	FACE ADVANCE = 12.1	0.19	0.38	20.30	79.13	0.0027	0.65	0.42
	FACE ADVANCE = 12.1	0.17	0.36	20.36	79.11	0.0023	0.58	0.40

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
30/10	FACE GB	0.08	0.17	20.75	79.00	0.0005	0.17	0.29
29/11	FACE GB	0.09	0.23	20.71	78.97	0.0005	0.20	0.25
02/12	OBE RETURN	0.13	0.27	20.64	78.96	0.0007	0.27	0.26
	10M RETURN RIP	0.06	0.21	20.53	79.10	0.0009	0.41	0.22
	FACE GB	0.08	0.18	20.75	78.99	0.0004	0.17	0.24
03/12	OBE RETURN	0.13	0.30	20.61	78.96	0.0007	0.30	0.23
	10M RETURN RIP	0.09	0.23	20.73	78.95	0.0003	0.18	0.17
	FACE GB	0.08	0.20	20.75	78.97	0.0003	0.26	0.19
	WASTE SAMPLES							
30/10	FACE ADVANCE = 15	0.17	0.50	20.03	79.30	0.0033	0.87	0.38
	FACE ADVANCE = 15	0.18	0.51	20.03	79.28	0.0033	0.87	0.38
29/11	FACE ADVANCE = 1	0.09	0.26	20.66	78.99	0.0008	0.26	0.31
02/12	FACE ADVANCE = 4	0.10	0.47	20.52	78.91	0.0010	0.37	0.27
	FACE ADVANCE = 4	0.10	0.45	20.54	78.91	0.0010	0.35	0.29

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
	WASTE SAMPLES							
03/12	FACE ADVANCE = 4	0.11	0.55	20.44	78.90	0.0013	0.45	0.29
	FACE ADVANCE = 4	0.11	0.56	20.42	78.91	0.0013	0.47	0.28

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
04/10	OBE RETURN	0.15	0.45	20.48	78.92	0.0007	0.42	0.17
	10M RETURN RIP	0.11	0.35	20.59	78.95	0.0004	0.31	0.13
	FACE GB	NO SAMPLE - VALVE REPLACED						
05/10	OBE RETURN	0.21	0.65	20.35	78.79	0.0009	0.52	0.17
	10M RETURN RIP	0.17	0.53	20.49	78.81	0.0005	0.38	0.13
	FACE GB	0.17	0.49	20.52	78.82	0.0004	0.36	0.11
06/12	OBE RETURN	NO SAMPLE - VALVE REPLACED						
	10M RETURN RIP	0.09	0.23	20.68	79.00	0.0004	0.24	0.17
	FACE GB	0.07	0.21	20.70	79.02	0.0003	0.23	0.13

DATE	SAMPLE LOCATION	CO2	CH4	O2	N2	CO	O2 Def	CO/Def
09/12	OBE RETURN	0.13	0.25	20.56	79.06	0.0009	0.37	0.24
	10M RETURN RIP	0.07	0.15	20.72	79.06	0.0004	0.21	0.19
	FACE GB	0.07	0.13	20.74	79.06	0.0004	0.19	0.21
	WASTE SAMPLES							
04/12	FACE ADVANCE = 7.5	0.05	1.10	19.80	79.05	0.0031	1.13	0.27
	FACE ADVANCE = 7.5	0.13	1.15	20.01	78.71	0.0023	0.83	0.28
05/12	FACE ADVANCE = 9.5	0.17	1.20	19.87	78.76	0.0027	0.99	0.27
	FACE ADVANCE = 9.5	0.17	1.15	19.90	78.78	0.0027	0.96	0.28
06/12	FACE ADVANCE = 10.5	0.15	0.75	20.05	79.05	0.0028	0.88	0.32
	FACE ADVANCE = 10.5	0.15	0.80	20.00	79.05	0.0029	0.93	0.31
09/12	FACE ADVANCE = 13.0	0.17	0.95	19.76	79.12	0.0042	1.19	0.35
	FACE ADVANCE = 13.0	0.17	0.95	19.75	79.13	0.0043	1.20	0.36

APPENDIX 2
FINITE ELEMENT PROGRAM INPUT FILE
FINITE ELEMENT PROGRAM
FINITE ELEMENT PROGRAM OUTPUT FILE

INPUT FILE

```
HARWORTH THREES
          166      150     5       0      0

0.18116E-01     0.18116E-01     0.00000E+00     0.00000E+00
 0.20130E-02    0.20130E-02     0.00000E+00     0.00000E+00
 0.22000E-03    0.22000E-03     0.00000E+00     0.00000E+00
 0.60380E-02    0.60380E-02     0.00000E+00     0.00000E+00
 0.54350E-03    0.54350E-03     0.00000E+00     0.00000E+00

    !X  Coordinate data !
0.00000E+00 .00000E+00 .00000E+00 .00000E+00 .00000E+00 .00000E+00
.00000E+00
0.00000E+00
0.00000E+00 .00000E+00 26.667    26.667    26.667    26.667
26.667    26.667
 26.667    26.667    26.667    26.667    53.333    53.333    53.333
53.333
 53.333    53.333    53.333    53.333    53.333    53.333    80.000
80.000
 80.000    80.000    80.000    80.000    80.000    80.000    80.000
80.000
 106.67    106.67    106.67    106.67    106.67    106.67    106.67
106.67
 106.67    106.67    133.33    133.33    133.33    133.33    133.33
133.33
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160.00
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186.67
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186.67
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293.33
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293.33
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320.00
 320.00    320.00    346.67    346.67    346.67    346.67    346.67
346.67
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380.00
 380.00    380.00    380.00    380.00    380.00    380.00    380.00
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380.00
 380.00    400.00    400.00    400.00    400.00    400.00    400.00
400.00
 400.00    400.00    400.00    400.00    400.00    400.00

!  Y coordinate data  !

 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    100.00
 150.00    200.00    230.00    240.00    250.00
 0.00000E+00 10.000    20.000    50.000    75.000
 100.00    125.00    150.00    175.00    200.00
 230.00    240.00    250.00    .00000E+00 10.000
 20.000    50.000    75.000    100.00    125.00
 150.00    175.00    200.00    230.00    240.00
 250.00

!    Element  Material Set     Nodes   !
          1        2      1   11   12    2
          2        1      2   12   13    3
          3        4      3   13   14    4
          4        2      4   14   15    5
          5        3      5   15   16    6
          6        2      6   16   17    7
          7        4      7   17   18    8
          8        1      8   18   19    9
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9	2	9	19	20	10
10	2	11	21	22	12
11	1	12	22	23	13
12	4	13	23	24	14
13	2	14	24	25	15
14	3	15	25	26	16
15	2	16	26	27	17
16	4	17	27	28	18
17	1	18	28	29	19
18	2	19	29	30	20
19	2	21	31	32	22
20	1	22	32	33	23
21	4	23	33	34	24
22	2	24	34	35	25
23	3	25	35	36	26
24	2	26	36	37	27
25	4	27	37	38	28
26	1	28	38	39	29
27	2	29	39	40	30
28	2	31	41	42	32
29	1	32	42	43	33
30	4	33	43	44	34
31	2	34	44	45	35
32	3	35	45	46	36
33	2	36	46	47	37
34	4	37	47	48	38
35	1	38	48	49	39
36	2	39	49	50	40
37	2	41	51	52	42
38	1	42	52	53	43
39	4	43	53	54	44
40	2	44	54	55	45
41	3	45	55	56	46
42	2	46	56	57	47
43	4	47	57	58	48
44	1	48	58	59	49
45	2	49	59	60	50
46	2	51	61	62	52
47	1	52	62	63	53
48	4	53	63	64	54
49	2	54	64	65	55
50	3	55	65	66	56
51	2	56	66	67	57
52	4	57	67	68	58
53	1	58	68	69	59
54	2	59	69	70	60
55	2	61	71	72	62
56	1	62	72	73	63
57	4	63	73	74	64
58	2	64	74	75	65
59	3	65	75	76	66
60	2	66	76	77	67
61	4	67	77	78	68
62	1	68	78	79	69

63	2	69	79	80	70
64	2	71	81	82	72
65	1	72	82	83	73
66	4	73	83	84	74
67	2	74	84	85	75
68	3	75	85	86	76
69	2	76	86	87	77
70	4	77	87	88	78
71	1	78	88	89	79
72	2	79	89	90	80
73	2	81	91	92	82
74	1	82	92	93	83
75	4	83	93	94	84
76	2	84	94	95	85
77	3	85	95	96	86
78	2	86	96	97	87
79	4	87	97	98	88
80	1	88	98	99	89
81	2	89	99	100	90
82	2	91	101	102	92
83	1	92	102	103	93
84	4	93	103	104	94
85	2	94	104	105	95
86	3	95	105	106	96
87	2	96	106	107	97
88	4	97	107	108	98
89	1	98	108	109	99
90	2	99	109	110	100
91	2	101	111	112	102
92	1	102	112	113	103
93	4	103	113	114	104
94	2	104	114	115	105
95	5	105	115	116	106
96	2	106	116	117	107
97	4	107	117	118	108
98	1	108	118	119	109
99	2	109	119	120	110
100	2	111	121	122	112
101	1	112	122	123	113
102	4	113	123	124	114
103	2	114	124	125	115
104	2	115	125	126	116
105	2	116	126	127	117
106	4	117	127	128	118
107	1	118	128	129	119
108	2	119	129	130	120
109	2	121	131	132	122
110	1	122	132	133	123
111	4	123	133	134	124
112	4	124	134	135	125
113	4	125	135	136	126
114	4	126	136	137	127
115	4	127	137	138	128
116	1	128	138	139	129

117	2	129	139	140	130
118	2	131	141	142	132
119	1	132	142	143	0
120	1	132	143	133	0
121	1	133	143	144	134
122	4	134	144	145	0
123	4	134	145	135	0
124	4	145	146	135	0
125	4	135	146	147	0
126	4	135	147	136	0
127	4	147	148	136	0
128	4	136	148	149	0
129	4	136	149	137	0
130	4	149	150	137	0
131	1	137	150	151	138
132	1	138	151	152	0
133	1	138	152	139	0
134	2	139	152	153	140
135	2	141	154	155	0
136	2	141	155	142	0
137	1	142	155	156	0
138	1	142	156	143	0
139	1	143	156	157	144
140	1	144	157	158	145
141	1	145	158	159	146
142	1	146	159	160	147
143	1	147	160	161	148
144	1	148	161	162	149
145	1	149	162	163	150
146	1	150	163	164	151
147	1	151	164	165	0
148	1	151	165	152	0
149	2	152	165	166	0
150	2	152	166	153	0

! Node	known pressure !
10	309.2
20	307.1
30	304.4
40	300.9
50	297.5
60	294.0
70	292.0
80	290.0
90	288.6
100	287.0
110	285.0
120	283.7
130	282.0
140	277.0
153	270.8
166	267.0

165	260.0
164	255.0
163	246.0
162	238.6
161	231.0
160	223.0
159	216.0
158	205.0
157	184.5
156	159.0
155	150.5
154	142.0
141	133.0
131	120.0
121	109.0
111	96.0
101	84.0
91	80.0
81	76.0
71	71.0
61	67.0
51	62.0
41	57.5
31	53.0
21	46.0
11	38.9
1	32.000
0	0.0

FINITE ELEMENT PROGRAM

```
      PROGRAM TDFIELD
      COMMON/ELMAT/X(4),Y(4),KL
      COMMON/ELMATX/ESM(4,4),EF(4)
      COMMON/MATL/DXE,DYE,GE,QE
      COMMON/HCV/IDBC(50,2),DBC(50,2),NDBC
      COMMON/AV/A(5000),JGF,JGSM,NP,NBW
      COMMON/TLE/TITLE(20)
      COMMON/SPLOT/XC(250),YC(250)
      COMMON/PLOTS/NE,NEL(300,4),NMTL(300),NCOEFF
      COMMON/CENT/XK(250),YK(250),GDX(4),GDY(4),GRADXY(250,2)
      COMMON/LOT/NN(250),GRDXY(250,2)
      COMMON/UNIT/IN,IO
      CHARACTER QU,INFILE*20,OUTPUT*20,GROUT*20
      INTEGER NS(4),ICK(250),Z
      REAL PHI(4),GDN(4)
      COMMON/D/DX(10),DY(10),G(10),Q(10)
      REAL B(3),C(3),GRDC(5,2)
      DATA GRDC /-1.,-1.,1.,-1.,-1.,1.,1.,1.,-1.,-1./
      DATA IFE/0/,VOL/0./,IS/5/,IR/6/
      IN=5
      IO=6
      DO 1, I=1,600
      A(I)=0.0
1     CONTINUE
      DO 201,I=1,10
      DX(I)=0
      DY(I)=0
      G(I)=0
      Q(I)=0
201   CONTINUE


210   WRITE(*,211)
211   FORMAT('1'////' WHAT IS THE INPUT FILE?')
      READ(*,212,ERR=210)INFILE
212   FORMAT(A20)

215   WRITE(*,213)
213   FORMAT('1'//' WHAT IS THE OUTPUT FILE?')
      READ(*,212,ERR=215)OUTPUT


      OPEN(UNIT=IN,FILE=INFILE,STATUS='OLD')
      OPEN(UNIT=IO,FILE=OUTPUT,STATUS='UNKNOWN')


C   DEFINITION OF INPUT PARAMETERS
C
C
```

```
C
C  TITLE - DESCRIPTIVE STATEMENT OF THE PROBLEM BEING SOLVED
C  NP - NUMBER OF EQUATIONS, ALSO NUMBER OF NODES
C  NE - NUMBER OF ELEMENTS
C  NCOEFF- NO. OF SETS OF EQUATION COEFFS. MAX 5
C
C*
C
C  DATA INPUT SECTION
C
C  TITLE CARD AND CONTROL PARAMETERS

      READ(IN,2) TITLE
2     FORMAT(20A4)
      READ (IN,*)NP,NE,NCOEFF,NDBC,IPLVL
      IF (NE.GT.10) THEN
      IPLVL=0
      ENDIF
      IF (IPLVL.GT.1) THEN
      IPLVL=0
      ENDIF
C COMPARISON OF NP, NE, NDBC, AND WITH THE
C   VALUES IN THE DIMENSION STATEMENTS

      ISTOP=0
C  CHECK NP
      IF(NP.GT.250) THEN
      WRITE(IO,3)
3       FORMAT(10X,'NUMBER OF NODES EXCEEDS 250 INPUT TERMINATED')
      ISTOP=1
      ENDIF
C  CHECK NE
      IF(NE.GT.300) THEN
        WRITE(IO,4)
4       FORMAT(10X,'NUMBER OF ELEMENTS EXCEEDS 300 INPUT
TERMINATED')
        ISTOP=1
      ENDIF
C CHECK NDBC
      IF(NDBC.GT.50) THEN
        WRITE(IO,5)
5       FORMAT(10X,'DERIVATIVE BOUNDARY CONDITION DATA',
     *'EXCEEDS DIMENSION OF 50'/10X,'INPUT TERMINATED')
        ISTOP=1
      ENDIF

C  CHECK OF NCOEFF

      IF (NCOEFF.GT.5) THEN
        WRITE(IO,7)
7       FORMAT(10X,'NCOEFF EXCEEDS 5  INPUT TERMINATED')
        ISTOP=1
      ENDIF
      Z=76
```

```
      IF(ISTOP.EQ.1) GOTO 990


C  INPUT OF EQUATION COEFFS AND NODAL COORDS

      READ(IN,*) (DX(I),DY(I),G(I),Q(I),I=1,NCOEFF)
      READ(IN,*) (XC(I),I=1,NP)
      READ(IN,*) (YC(I),I=1,NP)

C  OUTPUT OF TITLE AND PARAMETERS

       WRITE(IO,8) TITLE,NP,NE,IPLVL
8      FORMAT('1'///10X,20A4//10X,'NP = ',I5/10X,'NE = ',I5,/
     *10X,'IPLVL = ',I2)

C  OUTPUT OF THE EQUATION COEFFS

       WRITE(IO,9)
9      FORMAT(//10X,'EQUATION COEFFS',/10X,
     *'MATERIAL'/13X,'SET',8X,'DX',8X,'DY',13X,'G',14X,'Q')
       WRITE(IO,10) (I,DX(I),DY(I),G(I),Q(I),I=1,NCOEFF)
10     FORMAT(14X,I2,4G15.5E2)


C       OUTPUT OF NODAL COORDS

       WRITE(IO,11)
11     FORMAT(//10X,'NODAL COORDINATES'/10X,
     *'NODE',5X,'H',14X,'Y')
       WRITE(IO,12) (I,XC(I),YC(I),I=1,NP)
12     FORMAT(10X,I4,2G15.5E2)

C       INPUT AND ECHO PRINT OF ELEMENT NODAL DATA

       WRITE(IO,13) TITLE
13     FORMAT('1'///10X,20A4//10X,' ELEMENT DATA'/
     *15X,'NEL',4X,'NMTL',4X,'NODE NUMBERS')
       NID=0

       DO 15,KK=1,NE
       READ(IN,*) N,NMTL(KK),(NEL(N,I),I=1,4)
       IF((N-1).NE.NID) THEN
       WRITE(IO,14) N
14     FORMAT(10X,'ELEMENT',I4,' NOT IN SEQUENCE')
       Z=120
       GOTO 990
       ENDIF
       IF(NEL(N,4).EQ.0) THEN
         WRITE(IO,16)N,NMTL(KK),(NEL(N,I),I=1,3)
       ELSE
         WRITE(IO,16)N,NMTL(KK),(NEL(N,I),I=1,4)
       ENDIF
       NID = N
```

```
15     CONTINUE
16     FORMAT(15X,I3,5X,I3,2X,4I4)

       WRITE(*,202)
202    FORMAT(' ARE THERE ANY CHANGES TO ELEMENT PERMEABILITY CAUSED
BY
      $WASTE DISCONTINUITIES')
       READ(*,203) QU
       IF(QU.EQ.'N')GOTO 204
203    FORMAT(A1)
       CALL FAULTS
204    CONTINUE

C  INPUT AND ECHO PRINT OF THE DERIVATIVE BOUNDARY
C        CONDITION DATA

       IF (NDBC.NE.0)THEN
         WRITE(IO,17)
17       FORMAT(//10X,'DERIVATIVE BOUNDARY CONDITION DATA'/
      *15X,'ELEMENT',4X,'SIDE',7X,'ML',13X,'SL')
         DO 19, I=1,NDBC
         READ(IN,*)IDBC(I,1),IDBC(I,2),DBC(I,1),DBC(I,2)
         WRITE(IO,18) IDBC(I,1),IDBC(I,2),DBC(I,1),DBC(I,2)
18       FORMAT(15X,I4,9X,I1,2G15.5E2)
19       CONTINUE
       ENDIF

C  ANALYSIS OF THE NODE NUMBERS

C  INITIALIZATION OF A CHECK VECTOR

       DO 20, I=1,NP
       ICK(I)=0
20     CONTINUE

C   CHECK IF ANY NODE NUMBER EXCEEDS NP

       DO 23,I=1,NE
       KL=4
       IF(NEL(I,4).EQ.0) THEN
         KL=3
       ENDIF
       DO 22,J=1,KL
       K=NEL(I,J)
       ICK(K)=1
       IF(K.GT.NP) THEN
          WRITE(IO,21) J,I,NP
21        FORMAT(/10X,'NODE',I4,' OF ELEMENT',I4,
      *' EXCEEDS NP =',I4)
       Z=167
       GOTO  990
       ENDIF
```

```
22        CONTINUE
23        CONTINUE


C  CHECK TO SEE IF ALL NODE NUMBERS THROUGH
C         NP ARE INCLUDED

       DO 25, I=1,NP
       IF(ICK(I).EQ.0) THEN
          WRITE(IO,24) I
24        FORMAT(/10X,'NODE',I4,' DOES NOT EXIST')
          Z=181
          GOTO 990
       ENDIF
25        CONTINUE

       ZZ=AA
C  CREATE AND INITIALIZE THE A VECTOR

C   CALCULATE THE BANDWIDTH

       INBW=0
       NBW=0
       DO 29,KK=1,NE
       KL=4
       IF(NEL(KK,4).EQ.0) THEN
         KL=3
       ENDIF

       DO 26,I=1,KL
       NS(I)=NEL(KK,I)
 26    CONTINUE

       LK=KL-1
       DO 28,I=1,LK
       IJ=I+1
       DO 28,J=IJ,KL
       NB=IABS(NS(I)-NS(J))
       IF(NB.EQ.0)THEN
       WRITE(IO,27)KK
27       FORMAT(/10X,'ELEMENT',I3,' HAS TWO NODES WITH
      *THE SAME NODE NUMBER')
       Z=211
       GOTO 990
       ENDIF
       IF(NB.GT.NBW)THEN
       INBW=KK
       NBW=NB
       ENDIF
28      CONTINUE
29     CONTINUE
       NBW=NBW+1
       WRITE(IO,30)NBW,INBW
30       FORMAT(//10X,'BANDWIDTH IS',I4,' IN ELEMENT',I4)
```

```
C  CALCULATION OF POINTERS AND INITIALIZATION OF
C       COLUMN VECTOR A

        JGF=NP
        JGSM=JGF+NP
        JEND=JGSM+NP*NBW
        IF(JEND.GT.5000)THEN
        WRITE(IO,31)
31      FORMAT(10X,'DIMENSION OF A VECTOR EXCEEDED')
        STOP
        ENDIF
        DO 32,I=1,JEND
        A(I)=0.0
32      CONTINUE
        WRITE(IO,143)
C    GENERATION OF THE SYSTEM OF EQUATIONS
        READ(IN,123)
        DO 36,KK=1,NE
        KL=4
        IF(NEL(KK,4).EQ.0)THEN
        KL=3
        ENDIF

C  RETRIEVAL OF NODAL COORDS AND NODE NUMBERS
        DO 33,I=1,KL
        NS(I)=NEL(KK,I)
        J=NS(I)
        X(I)=XC(J)
        Y(I)=YC(J)
33        CONTINUE

C  ELEMENT COEFFS

        II=NMTL(KK)
        DXE=DX(II)
        DYE=DY(II)
        GE=G(II)
        QE=Q(II)

C  CALCULATION OF THE ELEMENT STIFFNESS MATRIX
C   AND ELEMENT FORCE VECTOR

        CALL ELSTMF(KK,IPLVL)

C  DIRECT STIFFNESS PROCEDURE

        DO 35,I=1,KL
        II=NS(I)
        A(JGF+II)=A(JGF+II)+EF(I)
        DO 34, J=1,KL
        JJ=NS(J)+1-II
        IF(JJ.GT.0) THEN
```

```
      J1=JGSM+(JJ-1)*NP+II-(JJ-1)*(JJ-2)/2
      A(J1)=A(J1)+ESM(I,J)
      ENDIF
34      CONTINUE
35      CONTINUE
36      CONTINUE


C  MODIFICATION AND SOLUTION OF THE SYSTEM OF EQUATIONS
C  OUTPUT OF THE CALCULATED NODAL VALUES

      WRITE(IO,37) TITLE
37      FORMAT('1'//10X,20A4)

C*************************************************************
      CALL MODIFY(IFE)
      CALL DCMPBD
      CALL SLVBD
      CALL PLOT
C*************************************************************


C  OUTPUT OF CALCULATED VALUES

      WRITE(IO,38)
38    FORMAT(//10X,'CALCULATED QUANTITIES',
     *12X,'NODAL VALUES FOR PHI')
      WRITE(IO,39) (I,A(I),I=1,NP)
39      FORMAT(12X,I3,G14.6E2,3X,I3,G14.6E2,3X,I3,G14.6E2)


C  EVALUATION OF VOLUME UNDER THE PHI SURFACE AND THE
C   ELEMENT GRADIENTS

C  START OF THE LOOP ON THE ELEMENTS

      ILINE=0
      DO 50,KK=1,NE

      IF(ILINE.LE.0) THEN

C OUTPUT OF THE CORRECT GRADIENT HEADING

        WRITE(IO,40) TITLE
40      FORMAT('1',///10X,20A4)

          WRITE(IO,42)
42        FORMAT(///10X,'ELEMENT',4X,'LOCATION',8X,'VEL(X)',
     *10X,'VEL(Y)')

      ENDIF
```

```
C   INCREASE THE LINE COUNT


        KL=4
        IF(NEL(KK,4).EQ.0) THEN
        KL=3
        ENDIF

        IF(KL.EQ.4) THEN
        ILINE=ILINE+4
        ENDIF

        IF(KL.EQ.3) THEN
        ILINE=ILINE+2
        ENDIF

        IF(ILINE.GT.50) THEN
        ILINE=0
        ENDIF


C   RETRIEVAL OF THE NODE NUMBERS, NODAL COORDS AND
C       NODAL VALUES OF PHI

        SP=0.0
        DO 44, I=1,KL
        NS(I)=NEL(KK,I)
        J=NS(I)
        X(I)=XC(J)
        Y(I)=YC(J)
        PHI(I)=A(J)
        SP=SP+PHI(I)
44      CONTINUE

        DO 441, I=1,250
        NN(I)=0
        GRDXY(I,1)=0.0
        GRDXY(I,2)=0.0
441     CONTINUE

C   ELEMENT COEFFS FOR THE GRADIENT VALUES

        II=NMTL(KK)
        DXE=DX(II)
        DYE=DY(II)

C   EVALUATION OF THE ELEMENT GRADIENTS

        IF(KL.NE.4) THEN

C TRIANGULAR ELEMENT

        B(1)=Y(2)-Y(3)
        B(2)=Y(3)-Y(1)
```

```
      B(3)=Y(1)-Y(2)
      C(1)=X(3)-X(2)
      C(2)=X(1)-X(3)
      C(3)=X(2)-X(1)

      AR2=X(2)*Y(3)+X(3)*Y(1)+X(1)*Y(2)
      -X(2)*Y(1)-X(3)*Y(2)-X(1)*Y(3)

      GRADX=(B(1)*PHI(1)+B(2)*PHI(2)+B(3)*PHI(3))/AR2
      GRADY=(C(1)*PHI(1)+C(2)*PHI(2)+C(3)*PHI(3))/AR2
      GDX(2)=DXE*GRADX*GRDC(4,1)
      GDY(2)=DYE*GRADY*GRDC(4,2)

      CALL CENTER(KK)


C OUTPUT OF GRADIENTS

      WRITE(IO,45) KK,GDX(2),GDY(2)

45    FORMAT(/13X,I3,5X,'CENTRE',2X,2G15.5E2)

C   CALCULATION OF THE VOLUME UNDER THE ELEMENT

      VOL =VOL+SP*AR2/6.0
      ENDIF

      IF(KL.EQ.4) THEN

C   RECTANGULAR ELEMENT

      AA=Y(4)-Y(1)
      BB=X(2)-X(1)
      AR=AA*BB

      GDX(1)=(PHI(2)-PHI(1))/BB
      GDX(2)=(-PHI(1)+PHI(2)+PHI(3)-PHI(4))/(2.0*BB)
      GDX(3)=(PHI(3)-PHI(4))/BB

      GDY(1)=(PHI(4)-PHI(1))/AA
      GDY(2)=(-PHI(1)-PHI(2)+PHI(3)+PHI(4))/(2.*AA)
      GDY(3)=(PHI(3)-PHI(2))/AA

      DO 46, I=1,3
      GDX(I)=DXE*GDX(I)*GRDC(4,1)
      GDY(I)=DYE*GDY(I)*GRDC(4,2)
46    CONTINUE

      CALL CENTER(KK)

C   THIS WRITES THE VELOCITIES AND NODE NUMBERS INTO ARRAYS
C     TO BE USED IN VPLOT WHEN THE RESULTANT VELOCITIES ARE
C        CALCULATED AND PLOTTED
```

```
      NA=NS(1)
      NB=NS(3)
      NN(NA)=NS(1)
      NN(NB)=NS(3)
      GRDXY(NA,1)=GDX(1)
      GRDXY(NA,2)=GDY(1)
      GRDXY(NB,1)=GDX(3)
      GRDXY(NB,2)=GDY(3)
```

```
C  OUTPUT FOR POTENTIAL FLOW, GROUNDWATER FLOW,
C     AND HEAT TRANSFER


      WRITE(IO,47) KK,NS(1),GDX(1),GDY(1)
      WRITE(IO,48) GDX(2),GDY(2)
      WRITE(IO,49) NS(3),GDX(3),GDY(3)

47      FORMAT(/13X,I3,5X,'NODE ',I3,2G15.5E2)
48      FORMAT(21X,'CENTRE',2X,2G15.5E2)
49      FORMAT(21X,'NODE ',I3,2G15.5E2)

C  CALCULATION OF THE VOLUME UNDER THE ELEMENT

      VOL=VOL+SP*AR/4.0
      ENDIF

50    CONTINUE
      CALL VPLOT(NP)

      GOTO 53
990   WRITE(IO,52) Z
52    FORMAT('  PROGRAM EXECUTION TERMINATED AT LINE ',I4)
53    STOP
      END


      SUBROUTINE ELSTMF(KK,IPLVL)
      COMMON/ELMAT/X(4),Y(4),KL
      COMMON/ELMATX/ESM(4,4),EF(4)
      COMMON/MATL/DXE,DYE,GE,QE
      COMMON/UNIT/IN,IO
      COMMON/HCV/IDBC(50,2),DBC(50,2),NDBC
      DIMENSION ES(4,4),ET(4,4),EG(4,4)
      DIMENSION B(3),C(3)
      REAL LG
      DATA ES/2.,-2.,-1.,1.,-2.,2.,1.,-1.,
     *-1.,1.,2.,-2.,1.,-1.,-2.,2./
      DATA ET/2.,1.,-1.,-2.,1.,2.,-2.,-1.,
     *-1.,-2.,2.,1.,-2.,-1.,1.,2./
      DATA EG/4.,2.,1.,2.,2.,4.,2.,1.,
```

```
     *1.,2.,4.,2.,2.,1.,2.,4./
      IF(KL.NE.4) THEN


C    LINEAR TRIANGULAR ELEMENT WITHOUT THE DERIVATIVE
C              BOUNDARY CONDITION


       B(1)=Y(2)-Y(3)
       B(2)=Y(3)-Y(1)
       B(3)=Y(1)-Y(2)
       C(1)=X(3)-X(2)
       C(2)=X(1)-X(3)
       C(3)=X(2)-X(1)

       AR2=X(2)*Y(3)+X(3)*Y(1)+X(1)*Y(2)-X(2)*Y(1)
     * -X(3)*Y(2)-X(1)*Y(3)

       IF(ABS(AR2).LT.0.0001) GOTO 109

       DO 101 I=1,3
       EF(I)=QE*AR2/6
       DO 101 J=1,3
       A=1.0
       IF(I.EQ.J) A=2.0
       ESM(I,J)=((DXE*B(I)*B(J)+DYE*C(I)*C(J))/(AR2*2.))
     *+A*GE*AR2/24.
101    CONTINUE
       IF(NDBC.EQ.0) GOTO 105
       GOTO 103
       ENDIF

C


C     BILINEAR RECTANGULAR ELEMENT WITHOUT THE
C         DERIVATIVE BOUNDARY CONDITION.


       AA=Y(4)-Y(1)
       BB=X(2)-X(1)
       AR=AA*BB
       IF(ABS(AR).LT.0.0001) GOTO 109

       DO 102 I=1,4
       EF(I)=QE*AR/4.0
       DO 102 J=1,4
       ESM(I,J)=DXE*AA*ES(I,J)/(6.0*BB)+DYE*BB*ET(I,J)/(6.0*AA)
       +GE*AR*EG(I,J)/36.0
102    CONTINUE
103    IF(NDBC.EQ.0) GOTO 105


C      DERIVATIVE BOUNDARY CONDITION
```

```
      DO 104 I=1,NDBC

      IF(IDBC(I,1).NE.KK) GOTO 104
       J=IDBC(I,2)
       K=J+1
       IF(J.EQ.KL) K=1
       EF(J)=EF(J)+DBC(I,2)/2.0
       EF(K)=EF(K)+DBC(I,2)/2.0
       ESM(J,J)=ESM(J,J)+DBC(I,1)/3.0
       ESM(J,K)=ESM(J,K)+DBC(I,1)/6.0
       ESM(K,J)=ESM(J,K)
       ESM(K,K)=ESM(K,K)+DBC(I,1)/3.0
104    CONTINUE


C    OUTPUT OF THE ELEMENT MATRICES


105    IF(IPLVL.EQ.0) RETURN
       WRITE(IO,106) KK
106     FORMAT(/10X,'ELEMENT',I4/10X,'FORCE VECTOR',10X,
     *'STIFFNESS MATRIX')

       DO 107 I=1,KL
       WRITE(IO,108) EF(I),(ESM(I,J),J=1,KL)
107    CONTINUE
108    FORMAT(10X,G12.5E2,10X,4G13.5E2)

       RETURN

C      DIAGNOSTIC OUTPUT


109    WRITE(IO,110) KK
110    FORMAT(//10X,'THE AREA OF ELEMENT',I4,
     *'IS LESS THAN 0.0001  THE NODE NUMBERS
     *ARE IN THE WRONG ORDER OR THE NODES
     *FORM A STRAIGHT LINE',/,
     *'EXECUTION TERMINATED')

       STOP
       END
```

```
       SUBROUTINE MODIFY(IFE)
       COMMON/AV/A(5000),JGF,JGSM,NP,NBW
       COMMON/UNIT/IN,IO
       CHARACTER QU
C
C    INPUT OF THE NODAL FORCE VALUES
C
C    FOR FIELD PROBLEMS
```

```
C         IB - NODE NUMBER
C         BV - SOURCE OR SINK VALUE
C
C
C
C
C
C
C   INPUT OF IB _BV TERMINATED BY INPUTTING
C         ZERO FOR IB AND BV

        NIW=0
        WRITE(*,212)
        READ(*,213)QU
212     FORMAT('1',////,' IS THERE ANY NITROGEN INJECTION?')
213     FORMAT(A1)
        IF(QU.EQ.'N') THEN
        GOTO 207
        ENDIF
201     WRITE(*,205)
        WRITE(IO,203)
202     READ(*,*,ERR=201) IB,BV
        IF(IB.LE.0) GOTO 207

203     FORMAT(//10X,'NITROGEN INJECTION QUANTITIES')
205     FORMAT('1'///' FOR NITROGEN INJECTION,  INPUT NODE NUMBER,
THEN
       $CUBIC M N2/SEC    ENTER 0    0.0    TO END')
        NIW=1
        A(JGF+IB)=A(JGF+IB)+BV
        WRITE(IO,206) IB,BV
206     FORMAT(10X,I3,G15.6E3)
        GOTO 202
C
C   INPUT OF THE PRESCRIBED NODAL VALUES
C    FOR FIELD PROBLEMS
C         IB - NODE NUMBER
C         BV - KNOWN VALUE OF PHI
C
C
C
C
C
C
C    INPUT TERMINATED BY ENTERING ZERO FOR IB AND BV

207     NIW=0
208     READ(IN,*) IB,BV
        IF(IB.LE.0) RETURN
        IF(NIW.EQ.0) WRITE(IO,210)
210     FORMAT(//10X,'KNOWN NODAL PRESSURE VALUES')
        NIW=1

C   MODIFICATION OF THE GLOBAL STIFFNESS MATRIX AND THE
C       GLOBAL FORCE VECTOR USING THE METHOD
```

```
C       OF DELETION OF ROWS AND COLUMNS

        K=IB-1
        DO 211 J=2,NBW

        M=IB+J-1
        IF(M.LE.NP) THEN
        IJ=JGSM+(J-1)*NP+IB-(J-1)*(J-2)/2
        A(JGF+M)=A(JGF+M)-A(IJ)*BV
        A(IJ)=0.0
        ENDIF

        IF(K.GT.0) THEN
        KJ=JGSM+(J-1)*NP+K-(J-1)*(J-2)/2
        A(JGF+K)=A(JGF+K)-A(KJ)*BV
        A(KJ)=0.0
        K=K-1
        ENDIF

211     CONTINUE
        A(JGF+IB)=A(JGSM+IB)*BV
221     CONTINUE

        WRITE(IO,206) IB,BV
        GOTO 208

        END
```

```
        SUBROUTINE DCMPBD
        COMMON/AV/A(5000),JGF,JGSM,NP,NBW
        COMMON/UNIT/IN,IO

C       DECOMPOSITION OF A BANDED MATRIX INTO AN UPPER
C       TRIANGULAR FORM

        NP1=NP-1

        DO 226,I=1,NP1
        MJ=I+NBW-1
        IF(MJ.GT.NP) MJ=NP
        NJ=I+1
        MK=NBW
        IF((NP-I+1).LT.NBW) MK=NP-I+1
        ND=0

        DO 225,J=NJ,MJ
        MK=MK-1
        ND=ND+1
        NL=ND+1
        DO 225,K=1,MK
        NK=ND+K
        JK=JGSM+(K-1)*NP+J-(K-1)*(K-2)/2
```

```
      INL=JGSM+(NL-1)*NP+I-(NL-1)*(NL-2)/2
      INK=JGSM+(NK-1)*NP+I-(NK-1)*(NK-2)/2
      II=JGSM+I
      A(JK)=A(JK)-(A(INL)/A(II))*A(INK)

225   CONTINUE
226   CONTINUE

      RETURN
      END
```

```
      SUBROUTINE SLVBD
      COMMON/AV/A(5000),JGF,JGSM,NP,NBW
      NP1=NP-1

C  DECOMPOSITION OF THE GLOBAL FOPRCE VECTOR


      DO 250, I=1,NP1
      MJ=I+NBW-1
      IF(MJ.GT.NP) MJ=NP
      NJ=I+1
      L=1

      DO 250, J=NJ,MJ
      L=L+1
      IL=JGSM+(L-1)*NP+I-(L-1)*(L-2)/2
      A(JGF+J)=A(JGF+J)-A(IL)*A(JGF+I)/A(JGSM+I)
250   CONTINUE

C  BACK SUBSTITUTION TO DETERMINE THE
C     NODAL VALUES

      A(NP)=A(JGF+NP)/A(JGSM+NP)
      DO 252, K=1,NP1
      I=NP-K
      MJ=NBW
      IF((I+NBW-1).GT.NP) MJ=NP-I+1
      SUM=0.0

      DO 251, J=2,MJ
      N=I+J-1
      IJ=JGSM+(J-1)*NP+I-(J-1)*(J-2)/2
      SUM=SUM+A(IJ)*A(N)
251   CONTINUE
      A(I)=(A(JGF+I)-SUM)/A(JGSM+I)
252   CONTINUE

      RETURN
      END
```

```
      SUBROUTINE PLOT
      COMMON/SPLOT/XC(250),YC(250)
      COMMON/PLOTS/NE,NEL(300,4),NMTL(300),NCOEFF
      COMMON/AV/A(5000),JGF,JGSM,NP,NBW
      DIMENSION IELEM(3,500)
      CHARACTER GTITLE*40

C     THIS DRAWS THE ELEMENT MESH AND CONTOURS
C         THE CALCULATED DATA

401   CONTINUE
      WRITE(*,402)
      READ(*,403,ERR=401) GTITLE
402   FORMAT('  INPUT THE UPPER GRAPH TITLE')
403   FORMAT(A40)



      CALL DIG
      CALL SIMBEG
      CALL PAGE(18.0,12.9)
      CALL PICSIZ(15.0,9.0)
      CALL NEWPIC

C    CALCULATES SCALES

      CALL LIMEXC(XC,NP,XMIN,XMAX)
      CALL LIMEXC(YC,NP,YMIN,YMAX)
      CALL SCALES(XMIN,XMAX,1,YMIN,YMAX,1)


C    DRAWS THE ELEMENT MESH

      DO 10, I=1,NE
      CALL BREAK
      IF(NEL(I,4).NE.0) THEN
      DO 1000,J=1,4
      CALL JOINPT(XC(NEL(I,J)),YC(NEL(I,J)))
1000  CONTINUE
      CALL JOINPT(XC(NEL(I,1)),YC(NEL(I,1)))
      CALL BREAK
      ELSE
      DO 1001, J=1,3
      CALL JOINPT(XC(NEL(I,J)),YC(NEL(I,J)))
1001  CONTINUE
      CALL JOINPT(XC(NEL(I,1)),YC(NEL(I,1)))
      CALL BREAK
      ENDIF
      CALL BREAK
10    CONTINUE


C  REORDERS THE POINTS TO SUIT SIMPLEPLOT, CALCULATES
C     CONTOUR SPACING AND DRAWS CONTOURS
```

```
CALL ZORDER(XC,YC,NP,IELEM,NELEM,500)
CALL LIMEXC(A,NP,AMIN,AMAX)
CALL SFEQZ((AMIN+(AMAX-AMIN)/10),(AMAX-AMIN)/10)
CALL CTNUMB(.TRUE.)
CALL ZCNTS(XC,YC,A,NP,IELEM,3,NELEM)
CALL SFLAB
CALL TEXTMG(0.5)
CALL TITLE('T','L','HIGH PRESSURE',13)
CALL TITLE('L','L','LOW PRESSURE',12)
CALL TEXTMG(1.0)
CALL TITLE('H','L','PRESSURE DISTRIBUTION',21)
CALL TITLE7('L','R',GTITLE)
RETURN
END



      SUBROUTINE CENTER(KK)
      COMMON/ELMAT/X(4),Y(4),KL
      COMMON/CENT/XK(250),YK(250),GDX(4),GDY(4),GRADXY(250,2)

C  CALCULATES THE CENTRAL COORDINATE OF EACH ELEMENT

      IF(KL.EQ.4) THEN
      XC = (X(1)+X(2))/2.0
      YC = (Y(1)+Y(4))/2.0

      ELSE
      XC = (X(1)+X(2)+X(3))/3.0
      YC = (Y(1)+Y(2)+Y(3))/3.0
      ENDIF

      XK(KK) = XC
      YK(KK) = YC
      GRADXY(KK,1) = GDX(2)
      GRADXY(KK,2) = GDY(2)

      RETURN
      END


      SUBROUTINE VPLOT(NP)
      COMMON/SPLOT/XC(250),YC(250)
      COMMON/PLOTS/NE,NEL(300,4),NMTL(300),NCOEFF
      COMMON/CENT/XK(250),YK(250),GDX(4),GDY(4),GRADXY(250,2)
      COMMON/LOT/NN(250),GRDXY(250,2)
      REAL GRAD(550)
      DIMENSION IELEM(3,500)
      CHARACTER GTITLE*50
C     PLOTS THE VELOCITIES, BUT FIRST.....

777   CONTINUE
      WRITE(*,778)
```

```
      READ(*,779,ERR=777) GTITLE
778   FORMAT('  INPUT THE LOWER GRAPH TITLE')
779   FORMAT(A50)



C       CALCULATES THE RESULTANT VELOCITY, WRITES THE DATA
C       INTO ARRAYS GRAD, FOR VELOCITY, AND XL _YL
      C=0.0
      DO 700, I=1,550
      GRAD(I)=C
700   CONTINUE

      C = 1.0E6
      DO 701, I=1,NE
      GRAD(I)=(SQRT((GRADXY(I,1)**2)+(GRADXY(I,2)**2)))*C
      GRAD(I)=(INT(GRAD(I)+0.5))/C

701   CONTINUE


      CALL PAGE(18.0,12.9)
      CALL PICSIZ(15.0,9.0)
      CALL NEWPIC
      CALL LIMEXC(XC,NP,XMIN,XMAX)
      CALL LIMEXC(YC,NP,YMIN,YMAX)
      CALL SCALES(XMIN,XMAX,1,YMIN,YMAX,1)
      CALL CTNUMB(.FALSE.)
C     DRAWS THE MESH

      DO 707, I=1,NE
      CALL BREAK
      IF(NEL(I,4).NE.0) THEN
      DO 705,J=1,4
      CALL JOINPT(XC(NEL(I,J)),YC(NEL(I,J)))
705   CONTINUE
      CALL JOINPT(XC(NEL(I,1)),YC(NEL(I,1)))
      CALL BREAK
      ELSE
      DO 706, J=1,3
      CALL JOINPT(XC(NEL(I,J)),YC(NEL(I,J)))
706   CONTINUE
      CALL JOINPT(XC(NEL(I,1)),YC(NEL(I,1)))
      CALL BREAK
      ENDIF
      CALL BREAK
707   CONTINUE


C  REORDERS THE MESH A LA SIMPLEPLOT
C   PLOTS THE VELOCITY CONTOURS

      CALL ZORDER(XK,YK,NE,IELEM,NELEM,500)
```

```
      CALL LIMEXC(GRAD,NE,GMIN,GMAX)
      CALL SFEQZ((GMIN+((GMAX-GMIN)/10)),((GMAX-GMIN)/10))
      CALL ZCNTS(XK,YK,GRAD,NE,IELEM,3,NELEM)
      CALL SFLAB
      CALL TEXTMG(0.5)
      CALL TITLE('T','L','HIGH PRESSURE',13)
      CALL TITLE('L','L','LOW PRESSURE',12)
      CALL TEXTMG(1.0)
      CALL TITLE('H','L','VELOCITY CONTOURS',17)
      CALL TITLE7('L','R',GTITLE)
      CALL ENDPLT
      WRITE(6,708)GMIN,GMAX
708   FORMAT('VELOCITY FROM',F8.7,'TO',F8.7)

      RETURN
      END


      SUBROUTINE FAULTS
      INTEGER N(300),NMT(300)
      CHARACTER QU
      COMMON/SPLOT/XC(250),YC(250)
      COMMON/PLOTS/NE,NEL(300,4),NMTL(300),NCOEFF
      COMMON/D/DX(10),DY(10),G(10),Q(10)
      COMMON/UNIT/IN,IO

      DO 801,I=1,NE
      NMT(I)=NMTL(I)
801   CONTINUE


      WRITE(*,*)' DO YOU WISH TO ASSIGN SPECIFIC PERMEABILITIES, N =
     $ACCEPT DEFAULT VALUES'
      READ(*,826) QU
      IF(QU.EQ.'Y') THEN
804   WRITE(*,*)' HOW MANY NEW VALUES,(MAX 5)'
      READ(*,*,ERR=804) I

      IF(I.GT.5) THEN
      WRITE(*,*)'  THAT IS MORE THAN 5 TRY AGAIN'
      GOTO 804
      ENDIF

      WRITE(*,*)'.INPUT NEW VALUES'
      WRITE(*,807)
807   FORMAT(//,' SET',6X,'DX',9X,'DY')

      DO 810,J=1,I
808   WRITE(*,*) J

      READ(*,*,ERR=808) DX(NCOEFF+J),DY(NCOEFF+J)
      IF(DX(NCOEFF+J).GT.1.) THEN
      WRITE(*,809)
809   FORMAT('  THAT IS TOO LARGE.  MAXIMUM IS 10-4')
```

```
      GOTO 808
      ELSEIF(DY(NCOEFF+J).GT.1.) THEN
      WRITE(*,809)
      GOTO 808
      ENDIF
810   CONTINUE

      WRITE(*,*)'  ENTER ELEMENT AFFECTED, AND NUMBER OF NEW MAT.
COEFF,
     $TWO ZEROS TO END.
     $IF YOU MAKE A MISTAKE, REEENTER THE ELEMENT AND MAT COEFF.'
      WRITE(*,*)'  IF YOU ENTER AN ELEMENT THAT SHOULD BE UNALTERED
THEN
     $ ENTER A NON-EXISTANT MAT COEFF.
     $IF YOU MAKE A MISTAKE AND ENTER THE PAIR, TOUGH'
      WRITE(*,811)
811   FORMAT(/,' ELEMENT     MAT COEFF')
      L=1

812   CONTINUE
      READ(*,*,ERR=812) N(L),MC

      IF(N(L).GT.NE) THEN
      WRITE(*,813)
813   FORMAT('  THE ELEMENT NUMBER EXCEEDS THE NUMBER
     $OF ELEMENTS.  TRY AGAIN')
      GOTO 812
      ELSEIF(MC.GT.I) THEN
      WRITE(*,*)'  THE MAT COEFF IS TOO LARGE.  TRY AGAIN'
      GOTO 812
      ELSEIF(N(L).EQ.0) THEN
      NCOEFF = NCOEFF + I
      GOTO 817
      ENDIF


      NMTL(N(L)) = MC + NCOEFF
      L = L + 1

      GOTO 812

      ELSE
      DO 814,I=NCOEFF+1,NCOEFF*2
      DX(I) = DX(I-NCOEFF)*5
      DY(I) = DY(I-NCOEFF)*5
814   CONTINUE
      WRITE(*,*)'  ENTER THE ELEMENTS AFFECTED. IF YOU ENTER A WRONG
     $ELEMENT THAT IS BAD NEWS.   ZERO TO END'
      L = 1
816   CONTINUE
      READ(*,*,ERR=816) N(L)
      IF(N(L).EQ.0) THEN
      NCOEFF = NCOEFF*2
      GOTO 817
```

```
      ELSEIF(N(L).GT.NE) THEN
      WRITE(*,813)
      GOTO 816
      ENDIF
      NMTL(N(L)) = NCOEFF + NMT(N(L))
      L=L+1
      GOTO 816
      ENDIF
C   OUTPUT OF ALTERED ELEMENT DATA
817   WRITE(IO,818)
818   FORMAT(///'  ELEMENTS AFFECTED BY DISCONTINUITIES')
      WRITE(IO,819)
819   FORMAT('  EQUATION COEFFICIENTS')
      WRITE(IO,820)
820   FORMAT('  SET',8X,'DX',15X,'DY')
      DO 821,I=1,NCOEFF
      WRITE(IO,822)I,DX(I),DY(I)
821   CONTINUE
822   FORMAT(3X,I2,9X,2G15.5E2)

      WRITE(IO,823)
823   FORMAT(//5X,'ELEMENT NUMBER',5X,'MATERIAL SET')
      DO 824,I=1,L
      IF(N(I).EQ.0) RETURN
      WRITE(IO,825) N(I),NMTL(N(I))
824   CONTINUE
825   FORMAT(12X,I3,15X,I3)
826   FORMAT(A1)

      RETURN
      END
```

OUTPUT FILE

HARWORTH THREES

NP = 166
NE = 150
IPLVL = 0

EQUATION COEFFS
MATERIAL

SET	DX	DY	G	Q
1	0.54350E-01	0.54350E-01	0.00000E+00	0.00000E+00
2	0.54350E-02	0.54350E-02	0.00000E+00	0.00000E+00
3	0.54350E-03	0.54350E-03	0.00000E+00	0.00000E+00
4	0.29270E-01	0.29270E-01	0.00000E+00	0.00000E+00
5	0.29270E-02	0.29270E-02	0.00000E+00	0.00000E+00

NODAL COORDINATES

NODE	X	Y
1	0.00000E+00	0.00000E+00
2	0.00000E+00	10.000
3	0.00000E+00	20.000
4	0.00000E+00	50.000
5	0.00000E+00	100.00
6	0.00000E+00	150.00
7	0.00000E+00	200.00
8	0.00000E+00	230.00
9	0.00000E+00	240.00
10	0.00000E+00	250.00
11	26.667	0.00000E+00
12	26.667	10.000
13	26.667	20.000
14	26.667	50.000
15	26.667	100.00
16	26.667	150.00
17	26.667	200.00
18	26.667	230.00
19	26.667	240.00
20	26.667	250.00
21	53.333	0.00000E+00
22	53.333	10.000
23	53.333	20.000
24	53.333	50.000
25	53.333	100.00
26	53.333	150.00
27	53.333	200.00
28	53.333	230.00
29	53.333	240.00

30	53.333	250.00
31	80.000	0.00000E+00
32	80.000	10.000
33	80.000	20.000
34	80.000	50.000
35	80.000	100.00
36	80.000	150.00
37	80.000	200.00
38	80.000	230.00
39	80.000	240.00
40	80.000	250.00
41	106.67	0.00000E+00
42	106.67	10.000
43	106.67	20.000
44	106.67	50.000
45	106.67	100.00
46	106.67	150.00
47	106.67	200.00
48	106.67	230.00
49	106.67	240.00
50	106.67	250.00
51	133.33	0.00000E+00
52	133.33	10.000
53	133.33	20.000
54	133.33	50.000
55	133.33	100.00
56	133.33	150.00
57	133.33	200.00
58	133.33	230.00
59	133.33	240.00
60	133.33	250.00
61	160.00	0.00000E+00
62	160.00	10.000
63	160.00	20.000
64	160.00	50.000
65	160.00	100.00
66	160.00	150.00
67	160.00	200.00
68	160.00	230.00
69	160.00	240.00
70	160.00	250.00
71	186.67	0.00000E+00
72	186.67	10.000
73	186.67	20.000
74	186.67	50.000
75	186.67	100.00
76	186.67	150.00
77	186.67	200.00
78	186.67	230.00
79	186.67	240.00
80	186.67	250.00
81	213.33	0.00000E+00
82	213.33	10.000
83	213.33	20.000

84	213.33	50.000
85	213.33	100.00
86	213.33	150.00
87	213.33	200.00
88	213.33	230.00
89	213.33	240.00
90	213.33	250.00
91	240.00	0.00000E+00
92	240.00	10.000
93	240.00	20.000
94	240.00	50.000
95	240.00	100.00
96	240.00	150.00
97	240.00	200.00
98	240.00	230.00
99	240.00	240.00
100	240.00	250.00
101	266.67	0.00000E+00
102	266.67	10.000
103	266.67	20.000
104	266.67	50.000
105	266.67	100.00
106	266.67	150.00
107	266.67	200.00
108	266.67	230.00
109	266.67	240.00
110	266.67	250.00
111	293.33	0.00000E+00
112	293.33	10.000
113	293.33	20.000
114	293.33	50.000
115	293.33	100.00
116	293.33	150.00
117	293.33	200.00
118	293.33	230.00
119	293.33	240.00
120	293.33	250.00
121	320.00	0.00000E+00
122	320.00	10.000
123	320.00	20.000
124	320.00	50.000
125	320.00	100.00
126	320.00	150.00
127	320.00	200.00
128	320.00	230.00
129	320.00	240.00
130	320.00	250.00
131	346.67	0.00000E+00
132	346.67	10.000
133	346.67	20.000
134	346.67	50.000
135	346.67	100.00
136	346.67	150.00
137	346.67	200.00

138	346.67	230.00
139	346.67	240.00
140	346.67	250.00
141	380.00	0.00000E+00
142	380.00	10.000
143	380.00	20.000
144	380.00	50.000
145	380.00	75.000
146	380.00	100.00
147	380.00	125.00
148	380.00	150.00
149	380.00	175.00
150	380.00	200.00
151	380.00	230.00
152	380.00	240.00
153	380.00	250.00
154	400.00	0.00000E+00
155	400.00	10.000
156	400.00	20.000
157	400.00	50.000
158	400.00	75.000
159	400.00	100.00
160	400.00	125.00
161	400.00	150.00
162	400.00	175.00
163	400.00	200.00
164	400.00	230.00
165	400.00	240.00
166	400.00	250.00

HARWORTH THREES

ELEMENT DATA

NEL	NMTL	NODE NUMBERS			
1	2	1	11	12	2
2	1	2	12	13	3
3	4	3	13	14	4
4	2	4	14	15	5
5	3	5	15	16	6
6	2	6	16	17	7
7	4	7	17	18	8
8	1	8	18	19	9
9	2	9	19	20	10
10	2	11	21	22	12
11	1	12	22	23	13
12	4	13	23	24	14
13	2	14	24	25	15
14	3	15	25	26	16
15	2	16	26	27	17
16	4	17	27	28	18
17	1	18	28	29	19
18	2	19	29	30	20

19	2	21	31	32	22
20	1	22	32	33	23
21	4	23	33	34	24
22	2	24	34	35	25
23	3	25	35	36	26
24	2	26	36	37	27
25	4	27	37	38	28
26	1	28	38	39	29
27	2	29	39	40	30
28	2	31	41	42	32
29	1	32	42	43	33
30	4	33	43	44	34
31	2	34	44	45	35
32	3	35	45	46	36
33	2	36	46	47	37
34	4	37	47	48	38
35	1	38	48	49	39
36	2	39	49	50	40
37	2	41	51	52	42
38	1	42	52	53	43
39	4	43	53	54	44
40	2	44	54	55	45
41	3	45	55	56	46
42	2	46	56	57	47
43	4	47	57	58	48
44	1	48	58	59	49
45	2	49	59	60	50
46	2	51	61	62	52
47	1	52	62	63	53
48	4	53	63	64	54
49	2	54	64	65	55
50	3	55	65	66	56
51	2	56	66	67	57
52	4	57	67	68	58
53	1	58	68	69	59
54	2	59	69	70	60
55	2	61	71	72	62
56	1	62	72	73	63
57	4	63	73	74	64
58	2	64	74	75	65
59	3	65	75	76	66
60	2	66	76	77	67
61	4	67	77	78	68
62	1	68	78	79	69
63	2	69	79	80	70
64	2	71	81	82	72
65	1	72	82	83	73
66	4	73	83	84	74
67	2	74	84	85	75
68	3	75	85	86	76
69	2	76	86	87	77
70	4	77	87	88	78
71	1	78	88	89	79
72	2	79	89	90	80

73	2	81	91	92	82
74	1	82	92	93	83
75	4	83	93	94	84
76	2	84	94	95	85
77	3	85	95	96	86
78	2	86	96	97	87
79	4	87	97	98	88
80	1	88	98	99	89
81	2	89	99	100	90
82	2	91	101	102	92
83	1	92	102	103	93
84	4	93	103	104	94
85	2	94	104	105	95
86.	3	95	105	106	96
87	2	96	106	107	97
88	4	97	107	108	98
89	1	98	108	109	99
90	2	99	109	110	100
91	2	101	111	112	102
92	1	102	112	113	103
93	4	103	113	114	104
94	2	104	114	115	105
95	5	105	115	116	106
96	2	106	116	117	107
97	4	107	117	118	108
98	1	108	118	119	109
99	2	109	119	120	110
100	2	111	121	122	112
101	1	112	122	123	113
102	4	113	123	124	114
103	2	114	124	125	115
104	2	115	125	126	116
105	2	116	126	127	117
106	4	117	127	128	118
107	1	118	128	129	119
108	2	119	129	130	120
109	2	121	131	132	122
110	1	122	132	133	123
111	4	123	133	134	124
112	4	124	134	135	125
113	4	125	135	136	126
114	4	126	136	137	127
115	4	127	137	138	128
116	1	128	138	139	129
117	2	129	139	140	130
118	2	131	141	142	132
119	1	132	142	143	
120	1	132	143	133	
121	1	133	143	144	134
122	4	134	144	145	
123	4	134	145	135	
124	4	145	146	135	
125	4	135	146	147	
126	4	135	147	136	

127	4	147	148	136	
128	4	136	148	149	
129	4	136	149	137	
130	4	149	150	137	
131	1	137	150	151	138
132	1	138	151	152	
133	1	138	152	139	
134	2	139	152	153	140
135	2	141	154	155	
136	2	141	155	142	
137	1	142	155	156	
138	1	142	156	143	
139	1	143	156	157	144
140	1	144	157	158	145
141	1	145	158	159	146
142	1	146	159	160	147
143	1	147	160	161	148
144	1	148	161	162	149
145	1	149	162	163	150
146	1	150	163	164	151
147	1	151	164	165	
148	1	151	165	152	
149	2	152	165	166	
150	2	152	166	153	

BANDWIDTH IS 15 IN ELEMENT 131

HARWORTH THREES

KNOWN	NODAL PRESSURE VALUES
10	309.200
20	307.100
30	304.400
40	300.900
50	297.500
60	294.000
70	292.000
80	290.000
90	288.600
100	287.000
110	285.000
120	283.700
130	282.000
140	277.000
153	270.800
166	267.000
165	260.000
164	255.000
163	246.000
162	238.600
161	231.000

160	223.000
159	216.000
158	205.000
157	184.500
156	159.000
155	150.500
154	142.000
141	133.000
131	120.000
121	109.000
111	96.0000
101	84.0000
91	80.0000
81	76.0000
71	71.0000
61	67.0000
51	62.0000
41	57.5000
31	53.0000
21	46.0000
11	38.9000
1	32.0000

NODE	PRESSURE	NODE	PRESSURE	NODE	PRESSURE
1	32.0000	2	48.8734	3	50.2159
4	54.5744	5	77.1680	6	275.231
7	296.180	8	299.311	9	300.097
10	309.200	11	38.9000	12	50.5421
13	51.5895	14	55.5252	15	77.7991
16	274.969	17	295.742	18	298.740
19	299.442	20	307.100	21	46.0000
22	54.1595	23	54.9083	24	58.1618
25	79.6285	26	274.217	27	294.502
28	297.246	29	297.851	30	304.400
31	53.0000	32	58.5984	33	59.1600
34	61.9433	35	82.5228	36	273.052
37	292.666	38	295.142	39	295.646
40	300.900	41	57.5000	42	63.2598
43	63.8071	44	66.4470	45	86.3287
46	271.570	47	290.457	48	292.700
49	293.126	50	297.500	51	62.0000
52	68.2493	53	68.8319	54	71.5201
55	90.9886	56	269.841	57	288.047
58	290.142	59	290.510	60	294.000
61	67.0000	62	73.7329	63	74.3724
64	77.2518	65	96.6534	66	267.846
67	285.498	68	287.600	69	287.994
70	292.000	71	71.0000	72	79.7992
73	80.5944	74	83.9027	75	103.787
76	265.406	77	282.747	78	284.995
79	285.442	80	290.000	81	76.0000
82	86.8355	83	87.8335	84	91.9052
85	113.428	86	262.007	87	279.589

88	282.165	89	282.720	90	288.600
91	80.0000	92	95.1076	93	96.4804
94	101.925	95	127.886	96	256.300
97	275.718	98	278.904	99	279.596
100	287.000	101	84.0000	102	105.539
103	107.350	104	114.149	105	153.137
106	244.375	107	271.161	108	274.982
109	275.824	110	285.000	111	96.0000
112	119.077	113	121.374	114	129.802
115	178.386	116	232.616	117	265.294
118	270.087	119	271.321	120	283.700
121	109.000	122	134.012	123	136.324
124	154.133	125	197.492	126	227.605
127	255.143	128	264.801	129	266.224
130	282.000	131	120.000	132	148.984
133	152.495	134	166.632	135	201.328
136	228.357	137	250.505	138	258.540
139	260.416	140	277.000	141	133.000
142	157.283	143	161.031	144	177.658
145	196.646	146	210.652	147	220.941
148	230.147	149	238.856	150	247.938
151	255.033	152	256.998	153	270.800
154	142.000	155	150.500	156	159.000
157	184.500	158	205.000	159	216.000
160	223.000	161	231.000	162	238.600
163	246.000	164	255.000	165	260.000
166	267.000				

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
1	NODE 1	-0.14063E-02	-0.91707E-02
	CENTRE	-0.87320E-03	-0.77491E-02
	NODE 12	-0.34010E-03	-0.63275E-02
2	NODE 2	-0.34010E-02	-0.72967E-02
	CENTRE	-0.31002E-02	-0.64946E-02
	NODE 13	-0.27994E-02	-0.56925E-02
	.		
3	NODE 3	-0.15076E-02	-0.42525E-02
	CENTRE	-0.12756E-02	-0.40462E-02
	NODE 14	-0.10436E-02	-0.38399E-02
4	NODE 4	-0.19377E-03	-0.24559E-02
	CENTRE	-0.16120E-03	-0.24386E-02
	NODE 15	-0.12863E-03	-0.24212E-02
5	NODE 5	-0.12863E-04	-0.21529E-02
	CENTRE	-0.37664E-05	-0.21481E-02

	NODE 16	0.53299E-05	-0.21432E-02
6	NODE 6	0.53299E-04	-0.22772E-02
	CENTRE	0.71309E-04	-0.22676E-02
	NODE 17	0.89319E-04	-0.22580E-02
7	NODE 7	0.48102E-03	-0.30548E-02
	CENTRE	0.55397E-03	-0.29900E-02
	NODE 18	0.62691E-03	-0.29251E-02
8	NODE 8	0.11641E-02	-0.42707E-02
	CENTRE	0.12489E-02	-0.40445E-02
	NODE 19	0.13338E-02	-0.38182E-02
9	NODE 9	0.13338E-03	-0.49476E-02
	CENTRE	0.28069E-03	-0.45547E-02
	NODE 20	0.42800E-03	-0.41619E-02
10	NODE 11	-0.14471E-02	-0.63275E-02
	CENTRE	-0.10922E-02	-0.53811E-02
	NODE 22	-0.73730E-03	-0.44347E-02
11	NODE 12	-0.73730E-02	-0.56925E-02
	CENTRE	-0.70687E-02	-0.48810E-02
	NODE 23	-0.67644E-02	-0:40696E-02
12	NODE 13	-0.36430E-02	-0.38399E-02
	CENTRE	-0.32685E-02	-0.35071E-02
	NODE 24	-0.28941E-02	-0.31743E-02
13	NODE 14	-0.53739E-03	-0.24212E-02
	CENTRE	-0.45512E-03	-0.23773E-02
	NODE 25	-0.37286E-03	-0.23334E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
14	NODE 15	-0.37286E-04	-0.21432E-02
	CENTRE	-0.10979E-04	-0.21292E-02
	NODE 26	0.15327E-04	-0.21152E-02
15	NODE 16	0.15327E-03	-0.22580E-02
	CENTRE	0.20298E-03	-0.22315E-02
	NODE 27	0.25270E-03	-0.22049E-02
16	NODE 17	0.13609E-02	-0.29251E-02
	CENTRE	0.15002E-02	-0.28013E-02
	NODE 28	0.16394E-02	-0.26775E-02

17	NODE 18	0.30442E-02	-0.38182E-02
	CENTRE	0.31444E-02	-0.35511E-02
	NODE 29	0.32445E-02	-0.32839E-02
18	NODE 19	0.32445E-03	-0.41619E-02
	CENTRE	0.43738E-03	-0.38608E-02
	NODE 30	0.55031E-03	-0.35596E-02
19	NODE 21	-0.14267E-02	-0.44347E-02
	CENTRE	-0.11657E-02	-0.37387E-02
	NODE 32	-0.90469E-03	-0.30427E-02
20	NODE 22	-0.90469E-02	-0.40696E-02
	CENTRE	-0.88561E-02	-0.35608E-02
	NODE 33	-0.86653E-02	-0.30520E-02
21	NODE 23	-0.46667E-02	-0.31743E-02
	CENTRE	-0.44086E-02	-0.29449E-02
	NODE 34	-0.41506E-02	-0.27156E-02
22	NODE 24	-0.77070E-03	-0.23334E-02
	CENTRE	-0.68030E-03	-0.22852E-02
	NODE 35	-0.58989E-03	-0.22370E-02
23	NODE 25	-0.58989E-04	-0.21152E-02
	CENTRE	-0.17617E-04	-0.20931E-02
	NODE 36	0.23754E-04	-0.20710E-02
24	NODE 26	0.23754E-03	-0.22049E-02
	CENTRE	0.30588E-03	-0.21685E-02
	NODE 37	0.37423E-03	-0.21320E-02
25	NODE 27	0.20154E-02	-0.26775E-02
	CENTRE	0.21628E-02	-0.25465E-02
	NODE 38	0.23102E-02	-0.24155E-02
26	NODE 28	0.42897E-02	-0.32839E-02
	CENTRE	0.43909E-02	-0.30141E-02
	NODE 39	0.44921E-02	-0.27443E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
27	NODE 29	0.44921E-03	-0.35596E-02
	CENTRE	0.58127E-03	-0.32075E-02
	NODE 40	0.71333E-03	-0.28553E-02
28	NODE 31	-0.91704E-03	-0.30427E-02
	CENTRE	-0.93349E-03	-0.30866E-02

	NODE 42	-0.94994E-03	-0.31305E-02
29	NODE 32	-0.94994E-02	-0.30520E-02
	CENTRE	-0.94848E-02	-0.30132E-02
	NODE 43	-0.94702E-02	-0.29743E-02
30	NODE 33	-0.51002E-02	-0.27156E-02
	CENTRE	-0.50215E-02	-0.26456E-02
	NODE 44	-0.49428E-02	-0.25757E-02
31	NODE 34	-0.91780E-03	-0.22370E-02
	CENTRE	-0.84669E-03	-0.21991E-02
	NODE 45	-0.77558E-03	-0.21611E-02
32	NODE 35	-0.77558E-04	-0.20710E-02
	CENTRE	-0.23683E-04	-0.20423E-02
	NODE 46	0.30191E-04	-0.20136E-02
33	NODE 36	0.30191E-03	-0.21320E-02
	CENTRE	0.37605E-03	-0.20925E-02
	NODE 47	0.45019E-03	-0.20530E-02
34	NODE 37	0.24245E-02	-0.24155E-02
	CENTRE	0.25522E-02	-0.23020E-02
	NODE 48	0.26799E-02	-0.21885E-02
35	NODE 38	0.49761E-02	-0.27443E-02
	CENTRE	0.50565E-02	-0.25298E-02
	NODE 49	0.51369E-02	-0.23153E-02
36	NODE 39	0.51369E-03	-0.28553E-02
	CENTRE	0.60328E-03	-0.26163E-02
	NODE 50	0.69288E-03	-0.23774E-02
37	NODE 41	-0.91739E-03	-0.31305E-02
	CENTRE	-0.96727E-03	-0.32635E-02
	NODE 52	-0.10172E-02	-0.33965E-02
38	NODE 42	-0.10172E-01	-0.29743E-02
	CENTRE	-0.10208E-01	-0.30705E-02
	NODE 53	-0.10244E-01	-0.31667E-02
39	NODE 43	-0.55168E-02	-0.25757E-02
	CENTRE	-0.55433E-02	-0.25992E-02
	NODE 54	-0.55698E-02	-0.26228E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
40	NODE 44	-0.10342E-02	-0.21611E-02

	CENTRE	-0.99211E-03	-0.21387E-02
	NODE 55	-0.94999E-03	-0.21162E-02
41	NODE 45	-0.94999E-04	-0.20136E-02
	CENTRE	-0.29868E-04	-0.19788E-02
	NODE 56	0.35262E-04	-0.19441E-02
42	NODE 46	0.35262E-03	-0.20530E-02
	CENTRE	0.42193E-03	-0.20160E-02
	NODE 57	0.49123E-03	-0.19790E-02
43	NODE 47	0.26455E-02	-0.21885E-02
	CENTRE	0.27268E-02	-0.21162E-02
	NODE 58	0.28081E-02	-0.20440E-02
44	NODE 48	0.52142E-02	-0.23153E-02
	CENTRE	0.52738E-02	-0.21563E-02
	NODE 59	0.53335E-02	-0.19972E-02
45	NODE 49	0.53335E-03	-0.23774E-02
	CENTRE	0.62344E-03	-0.21372E-02
	NODE 60	0.71352E-03	-0.18971E-02
46	NODE 51	-0.10189E-02	-0.33965E-02
	CENTRE	-0.10682E-02	-0.35279E-02
	NODE 62	-0.11175E-02	-0.36593E-02
47	NODE 52	-0.11175E-01	-0.31667E-02
	CENTRE	-0.11233E-01	-0.33213E-02
	NODE 63	-0.11291E-01	-0.34759E-02
48	NODE 53	-0.60806E-02	-0.26228E-02
	CENTRE	-0.61855E-02	-0.27161E-02
	NODE 64	-0.62905E-02	-0.28093E-02
49	NODE 54	-0.11680E-02	-0.21162E-02
	CENTRE	-0.11612E-02	-0.21126E-02
	NODE 65	-0.11544E-02	-0.21089E-02
50	NODE 55	-0.11544E-03	-0.19441E-02
	CENTRE	-0.37392E-04	-0.19025E-02
	NODE 66	0.40656E-04	-0.18609E-02
51	NODE 56	0.40656E-03	-0.19790E-02
	CENTRE	0.46300E-03	-0.19489E-02
	NODE 67	0.51944E-03	-0.19188E-02
52	NODE 57	0.27974E-02	-0.20440E-02
	CENTRE	0.27934E-02	-0.20476E-02
	NODE 68	0.27894E-02	-0.20512E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
53	NODE 58	0.51795E-02	-0.19972E-02
	CENTRE	0.51532E-02	-0.20671E-02
	NODE 69	0.51270E-02	-0.21371E-02
54	NODE 59	0.51270E-03	-0.18971E-02
	CENTRE	0.46014E-03	-0.20372E-02
	NODE 70	0.40757E-03	-0.21774E-02
55	NODE 61	-0.81515E-03	-0.36593E-02
	CENTRE	-0.10257E-02	-0.42208E-02
	NODE 72	-0.12362E-02	-0.47824E-02
56	NODE 62	-0.12362E-01	-0.34759E-02
	CENTRE	-0.12521E-01	-0.38988E-02
	NODE 73	-0.12680E-01	-0.43217E-02
57	NODE 63	-0.68285E-02	-0.28093E-02
	CENTRE	-0.70639E-02	-0.30186E-02
	NODE 74	-0.72993E-02	-0.32278E-02
58	NODE 64	-0.13554E-02	-0.21089E-02
	CENTRE	-0.14046E-02	-0.21352E-02
	NODE 75	-0.14538E-02	-0.21615E-02
59	NODE 65	-0.14538E-03	-0.18609E-02
	CENTRE	-0.47829E-04	-0.18088E-02
	NODE 76	0.49725E-04	-0.17568E-02
60	NODE 66	0.49725E-03	-0.19188E-02
	CENTRE	0.52892E-03	-0.19019E-02
	NODE 77	0.56060E-03	-0.18850E-02
61	NODE 67	0.30191E-02	-0.20512E-02
	CENTRE	0.29394E-02	-0.21220E-02
	NODE 78	0.28597E-02	-0.21928E-02
62	NODE 68	0.53101E-02	-0.21371E-02
	CENTRE	0.52553E-02	-0.22830E-02
	NODE 79	0.52006E-02	-0.24289E-02
63	NODE 69	0.52006E-03	-0.21774E-02
	CENTRE	0.46382E-03	-0.23274E-02
	NODE 80	0.40757E-03	-0.24774E-02
64	NODE 71	-0.10193E-02	-0.47824E-02
	CENTRE	-0.12269E-02	-0.53357E-02
	NODE 82	-0.14344E-02	-0.58891E-02
65	NODE 72	-0.14344E-01	-0.43217E-02

	CENTRE	-0.14551E-01	-0.48729E-02
	NODE 83	-0.14758E-01	-0.54241E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
66	NODE 73	-0.79478E-02	-0.32278E-02
	CENTRE	-0.83669E-02	-0.36003E-02
	NODE 84	-0.87859E-02	-0.39727E-02
67	NODE 74	-0.16314E-02	-0.21615E-02
	CENTRE	-0.17984E-02	-0.22505E-02
	NODE 85	-0.19654E-02	-0.23395E-02
68	NODE 75	-0.19654E-03	-0.17568E-02
	CENTRE	-0.63626E-04	-0.16859E-02
	NODE 86	0.69285E-04	-0.16151E-02
69	NODE 76	0.69285E-03	-0.18850E-02
	CENTRE	0.66835E-03	-0.18981E-02
	NODE 87	0.64386E-03	-0.19112E-02
70	NODE 77	0.34675E-02	-0.21928E-02
	CENTRE	0.32870E-02	-0.23532E-02
	NODE 88	0.31066E-02	-0.25135E-02
71	NODE 78	0.57685E-02	-0.24289E-02
	CENTRE	0.56581E-02	-0.27232E-02
	NODE 89	0.55478E-02	-0.30174E-02
72	NODE 79	0.55478E-03	-0.24774E-02
	CENTRE	0.42009E-03	-0.28365E-02
	NODE 90	0.28541E-03	-0.31956E-02
73	NODE 81	-0.81515E-03	-0.58891E-02
	CENTRE	-0.12505E-02	-0.70500E-02
	NODE 92	-0.16858E-02	-0.82110E-02
74	NODE 82	-0.16858E-01	-0.54241E-02
	CENTRE	-0.17239E-01	-0.64425E-02
	NODE 93	-0.17621E-01	-0.74609E-02
75	NODE 83	-0.94899E-02	-0.39727E-02
	CENTRE	-0.10244E-01	-0.46426E-02
	NODE 94	-0.10997E-01	-0.53125E-02
76	NODE 84	-0.20420E-02	-0.23395E-02
	CENTRE	-0.24942E-02	-0.25807E-02
	NODE 95	-0.29464E-02	-0.28220E-02

77	NODE 85	-0.29464E-03	-0.16151E-02
	CENTRE	-0.89168E-04	-0.15055E-02
	NODE 96	0.11631E-03	-0.13959E-02
78	NODE 86	0.11631E-02	-0.19112E-02
	CENTRE	0.97597E-03	-0.20110E-02
	NODE 97	0.78886E-03	-0.21108E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
79	NODE 87	0.42484E-02	-0.25135E-02
	CENTRE	0.39140E-02	-0.28108E-02
	NODE 98	0.35796E-02	-0.31081E-02
80	NODE 88	0.66468E-02	-0.30174E-02
	CENTRE	0.65066E-02	-0.33913E-02
	NODE 99	0.63664E-02	-0.37652E-02
81	NODE 89	0.63664E-03	-0.31956E-02
	CENTRE	0.48135E-03	-0.36097E-02
	NODE 100	0.32606E-03	-0.40239E-02
82	NODE 91	-0.81515E-03	-0.82110E-02
	CENTRE	-0.14705E-02	-0.99587E-02
	NODE 102	-0.21258E-02	-0.11706E-01
83	NODE 92	-0.21258E-01	-0.74609E-02
	CENTRE	-0.21705E-01	-0.86523E-02
	NODE 103	-0.22151E-01	-0.98436E-02
84	NODE 93	-0.11929E-01	-0.53125E-02
	CENTRE	-0.12672E-01	-0.59730E-02
	NODE 104	-0.13415E-01	-0.66334E-02
85	NODE 94	-0.24910E-02	-0.28220E-02
	CENTRE	-0.38184E-02	-0.35300E-02
	NODE 105	-0.51457E-02	-0.42380E-02
86	NODE 95	-0.51457E-03	-0.13959E-02
	CENTRE	-0.13578E-03	-0.11938E-02
	NODE 106	0.24301E-03	-0.99175E-03
87	NODE 96	0.24301E-02	-0.21108E-02
	CENTRE	0.16794E-02	-0.25112E-02
	NODE 107	0.92865E-03	-0.29117E-02
88	NODE 97	0.50012E-02	-0.31081E-02

	CENTRE	0.46525E-02	-0.34181E-02
	NODE 108	0.43038E-02	-0.37281E-02
89	NODE 98	0.79915E-02	-0.37652E-02
	CENTRE	0.78395E-02	-0.41704E-02
	NODE 109	0.76876E-02	-0.45757E-02
90	NODE 99	0.76876E-03	-0.40239E-02
	CENTRE	0.58817E-03	-0.45055E-02
	NODE 110	0.40757E-03	-0.49872E-02
91	NODE 101	-0.24464E-02	-0.11706E-01
	CENTRE	-0.26031E-02	-0.12124E-01
	NODE 112	-0.27599E-02	-0.12542E-01

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
92	NODE 102	-0.27599E-01	-0.98436E-02
	CENTRE	-0.28094E-01	-0.11165E-01
	NODE 113	-0.28590E-01	-0.12486E-01
93	NODE 103	-0.15397E-01	-0.66334E-02
	CENTRE	-0.16291E-01	-0.74280E-02
	NODE 114	-0.17185E-01	-0.82227E-02
94	NODE 104	-0.31911E-02	-0.42380E-02
	CENTRE	-0.41693E-02	-0.47595E-02
	NODE 115	-0.51474E-02	-0.52811E-02
95	NODE 105	-0.27721E-02	-0.53411E-02
	CENTRE	-0.74055E-03	-0.42578E-02
	NODE 116	0.12910E-02	-0.31746E-02
96	NODE 106	0.23972E-02	-0.29117E-02
	CENTRE	0.17967E-02	-0.32319E-02
	NODE 117	0.11961E-02	-0.35521E-02
97	NODE 107	0.64416E-02	-0.37281E-02
	CENTRE	0.59081E-02	-0.42022E-02
	NODE 118	0.53746E-02	-0.46763E-02
98	NODE 108	0.99798E-02	-0.45757E-02
	CENTRE	0.95795E-02	-0.56427E-02
	NODE 119	0.91793E-02	-0.67097E-02
99	NODE 109	0.91793E-03	-0.49872E-02
	CENTRE	0.59148E-03	-0.58575E-02
	NODE 120	0.26502E-03	-0.67278E-02

100	NODE 111	-0.26492E-02	-0.12542E-01
	CENTRE	-0.28464E-02	-0.13068E-01
	NODE 122	-0.30435E-02	-0.13594E-01
101	NODE 112	-0.30435E-01	-0.12486E-01
	CENTRE	-0.30450E-01	-0.12525E-01
	NODE 123	-0.30465E-01	-0.12564E-01
102	NODE 113	-0.16407E-01	-0.82227E-02
	CENTRE	-0.21555E-01	-0.12799E-01
	NODE 124	-0.26703E-01	-0.17376E-01
103	NODE 114	-0.49583E-02	-0.52811E-02
	CENTRE	-0.44258E-02	-0.49971E-02
	NODE 125	-0.38934E-02	-0.47131E-02
104	NODE 115	-0.38934E-02	-0.58947E-02
	CENTRE	-0.14362E-02	-0.45840E-02
	NODE 126	0.10210E-02	-0.32734E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
105	NODE 116	0.10210E-02	-0.35521E-02
	CENTRE	0.15448E-02	-0.32727E-02
	NODE 127	0.20686E-02	-0.29934E-02
106	NODE 117	0.11140E-01	-0.46763E-02
	CENTRE	0.84705E-02	-0.70496E-02
	NODE 128	0.58009E-02	-0.94230E-02
107	NODE 118	0.10771E-01	-0.67097E-02
	CENTRE	0.10579E-01	-0.72220E-02
	NODE 129	0.10387E-01	-0.77343E-02
108	NODE 119	0.10387E-02	-0.67278E-02
	CENTRE	0.69258E-03	-0.76510E-02
	NODE 130	0.34644E-03	-0.85741E-02
109	NODE 121	-0.22417E-02	-0.13594E-01
	CENTRE	-0.26463E-02	-0.14673E-01
	NODE 132	-0.30510E-02	-0.15753E-01
110	NODE 122	-0.30510E-01	-0.12564E-01
	CENTRE	-0.31733E-01	-0.15825E-01
	NODE 133	-0.32956E-01	-0.19087E-01
111	NODE 123	-0.17748E-01	-0.17376E-01
	CENTRE	-0.15733E-01	-0.15584E-01
	NODE 134	-0.13718E-01	-0.13792E-01

112	NODE 124	-0.13718E-01	-0.25382E-01
	CENTRE	-0.89641E-02	-0.22847E-01
	NODE 135	-0.42105E-02	-0.20311E-01
113	NODE 125	-0.42105E-02	-0.17629E-01
	CENTRE	-0.25179E-02	-0.16726E-01
	NODE 136	-0.82540E-03	-0.15823E-01
114	NODE 126	-0.82540E-03	-0.16121E-01
	CENTRE	0.21325E-02	-0.14543E-01
	NODE 137	0.50903E-02	-0.12965E-01
115	NODE 127	0.50903E-02	-0.94230E-02
	CENTRE	0.59809E-02	-0.86312E-02
	NODE 138	0.68715E-02	-0.78395E-02
116	NODE 128	0.12759E-01	-0.77343E-02
	CENTRE	0.12297E-01	-0.89662E-02
	NODE 139	0.11836E-01	-0.10198E-01
117	NODE 129	0.11836E-02	-0.85741E-02
	CENTRE	0.11012E-02	-0.87937E-02
	NODE 140	0.10189E-02	-0.90132E-02

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
118	NODE 131	-0.21199E-02	-0.15753E-01
	CENTRE	-0.17366E-02	-0.14475E-01
	NODE 142	-0.13534E-02	-0.13198E-01
119	CENTRE	-0.13534E-01	-0.20371E-01
120	CENTRE	-0.13919E-01	-0.19087E-01
121	NODE 133	-0.13919E-01	-0.25610E-01
	CENTRE	-0.15949E-01	-0.27866E-01
	NODE 144	-0.17979E-01	-0.30122E-01
122	CENTRE	-0.96827E-02	-0.22232E-01
123	CENTRE	-0.11124E-01	-0.20311E-01
124	CENTRE	-0.81885E-02	-0.16398E-01
125	CENTRE	-0.81885E-02	-0.12046E-01
126	CENTRE	-0.53555E-02	-0.15823E-01

127	CENTRE	-0.15714E-02	-0.10778E-01
128	CENTRE	-0.15714E-02	-0.10197E-01
129	CENTRE	0.50528E-03	-0.12965E-01
130	CENTRE	0.22544E-02	-0.10633E-01
131	NODE 137	0.41861E-02	-0.14557E-01
	CENTRE	0.49522E-02	-0.13706E-01
	NODE 151	0.57183E-02	-0.12855E-01
132	CENTRE	0.57183E-02	-0.10677E-01
133	CENTRE	0.55745E-02	-0.10198E-01
134	NODE 139	0.55745E-03	-0.90132E-02
	CENTRE	0.78423E-03	-0.82573E-02
	NODE 153	0.10110E-02	-0.75015E-02
135	CENTRE	-0.24458E-02	-0.46197E-02
136	CENTRE	0.18433E-02	-0.13198E-01
137	CENTRE	0.18433E-01	-0.46197E-01
138	CENTRE	0.55200E-02	-0.20371E-01
139	NODE 143	0.55200E-02	-0.30122E-01
	CENTRE	-0.65370E-02	-0.38160E-01
	NODE 157	-0.18594E-01	-0.46198E-01

HARWORTH THREES

ELEMENT	LOCATION	VEL(X)	VEL(Y)
140	NODE 144	-0.18594E-01	-0.41282E-01
	CENTRE	-0.20647E-01	-0.42924E-01
	NODE 158	-0.22701E-01	-0.44567E-01
141	NODE 145	-0.22701E-01	-0.30449E-01
	CENTRE	-0.18617E-01	-0.27181E-01
	NODE 159	-0.14532E-01	-0.23914E-01
142	NODE 146	-0.14532E-01	-0.22368E-01
	CENTRE	-0.10064E-01	-0.18793E-01
	NODE 160	-0.55953E-02	-0.15218E-01
143	NODE 147	-0.55953E-02	-0.20013E-01

	CENTRE	-0.39571E-02	-0.18703E-01
	NODE 161	-0.23190E-02	-0.17392E-01
144	NODE 148	-0.23190E-02	-0.18934E-01
	CENTRE	-0.81192E-03	-0.17728E-01
	NODE 162	0.69511E-03	-0.16522E-01
145	NODE 149	0.69511E-03	-0.19744E-01
	CENTRE	0.29807E-02	-0.17916E-01
	NODE 163	0.52662E-02	-0.16088E-01
146	NODE 150	0.52662E-02	-0.12855E-01
	CENTRE	0.26784E-02	-0.14580E-01
	NODE 164	0.90586E-04	-0.16305E-01
147	CENTRE	0.90586E-04	-0.27175E-01
148	CENTRE	-0.81583E-02	-0.10677E-01
149	CENTRE	-0.81583E-03	-0.38045E-02
150	CENTRE	0.10327E-02	-0.75015E-02